PRELIMINARY ECONOMIC ASSESSMENT FOR THE KOPSA COPPER-GOLD DEPOSIT, FINLAND

Prepared For

BELVEDERE RESOURCES LTD

Report Prepared by

srk consulting

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SE443

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EXECUTIVE SUMMARY
PRELIMINARY ECONOMIC ASSESSMENT FOR THE KOPSA COPPER-GOLD DEPOSIT, FINLAND

1. SUMMARY

1.1 Introduction

This report has been prepared for Belvedere Resources Ltd (Belvedere) and made compliant with the Canadian Institute of Mining, Metallurgy and Petroleum (CIM) definitions and guidelines of National Instrument 43-101 and accompanying documents 43-101.F1 and 43-101.CP. The author's scope of work for this document has been to produce a preliminary economic assessment (PEA) for the Kopsa deposit, which is 100% owned by Belvedere.

The Project is at a conceptual stage but it is currently envisaged that it will comprise a single open pit mine at the Kopsa site, with on-site crushing and possible sorting based on X-ray transmission (XRT) technology. Material will then be trucked to the Company's existing processing facility at Hitura for production of a marketable copper sulphide concentrate and smelted gold/silver doré through conventional flotation, cyanide leaching and Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL). Subject to financing, the Company expects to commence a feasibility study in Q4 2013.

1.2 Geology, Data Quality and Resource Estimation

The Kopsa deposit comprises several bodies of Cu-Au mineralization tentatively classed as an intrusive-related style of deposit. Within a low grade halo of mineralization, veins containing elevated Au grades are present. SRK created a geological model based on a statistical review of the validated drillhole data. Two domains were outlined by SRK – an Au-rich and a Cu-rich domain. These domains were created based on statistical grade breaks with a 0.08% Cu, and 0.2 g/t Au cut-off being utilised to delineate the domains. It was not possible to model the individual high-grade Au veins due to the current drill spacing and nature of the mineralization.

The data used in the estimation and the associated quality control quality assurance (QAQC) data was provided to SRK by Belvedere. It is the opinion of SRK that the results of the blanks, certified reference materials, and the results of duplicated samples show that a reasonable level of confidence can be attributed to the recent drill samples used in the Mineral Resource estimate.

A 2 m composite file was used in a geostatistical study (variography and quantitative kriging neighbourhood analysis - QKNA) that resulted in ordinary kriging (OK) being selected as the interpolation method. The interpolation used an elliptical search following the predominant dip and dip direction of the geological domains. The results of the variography and the QKNA were utilised to determine the most appropriate search parameters.

The interpolated block model was validated through visual checks, a comparison of the mean composite and block grades and through the generation of section validation slices. SRK is confident that the interpolated grades are a reasonable reflection of the available sample data.

1.3 Mineral Resources

Table ES 1 below presents a Mineral Resource statement for the Kopsa deposit. A pit optimisation exercise was carried out based on assumed operating costs, slope angles, mining recoveries and revenue assumptions derived from, SRK's experience and was used to constrain the Mineral Resource statement to that material which SRK considers has reasonable prospect for eventual economic extraction.

The statement has been classified in accordance with the CIM Definitions by the QP, Lucy Roberts (MAusIMM(CP)), who is an independent consultant with no relationship to Belvedere and has never been employed by Belvedere. It has an effective date of 02 October 2013.

The quantity and grade of reported Inferred Resources in this estimation are uncertain in nature and there has been insufficient exploration to define these as an Indicated or Measured Mineral Resource. It is uncertain if further exploration will result in upgrading these to an Indicated or Measured Mineral Resource category.

SRK is not aware of any environmental, permitting, legal, title, taxation, socio-political, marketing, or other relevant issues that would preclude the report of the Mineral Resource given here.

Table ES 1: Kopsa Mineral Resource statement

Category	Tonnes (Mt)	Au (g/t)	Cu (%)	AuEq (g/t)	Ag (g/t)
Measured	11.5	0.83	0.15	1.07	2.17
Indicated	2.2	0.70	0.15	0.95	2.08
Measured+Indicated	**13.6**	**0.81**	**0.15**	**1.05**	**2.15**
Inferred	2.7	0.8	0.2	1.1	2.57

(1) The effective date of the Mineral Resource statement is 02 October 2013.

(2) The Mineral Resource reported for Kopsa was constrained within a Lerchs-Grossman pit shell defined by a marginal cut-off-grade of 0.50 g/t AuEq, a metal price for copper USD7870 / t and metal price for gold USD1508 / oz; overall slope angles of 45° and 50° for the footwall and hangingwall correspondingly; a mining recovery of 97%; a mining dilution of 5%; mining costs of USD3.5/tonne, process operating costs (inclusive of G&A costs) of USD13/tonne material processed; and transport costs of USD5.6/tonne.

*(3) Gold Equivalent (AuEq) (g/t) = 0.982830*Au (g/t) + 1.672011*Cu (%)*

1.4 Mining Methods

SRK has evaluated the potential to mine the deposit using an open pit mining method and reviewed the available geotechnical and hydrogeological information to determine suitable slope angle. SRK has produced a preliminary production schedule and estimated appropriate mining costs.

A conventional approach to open pit mining using an excavator-truck configuration is proposed for mining. A conceptual production rate of 1.2 Mtpa is considered appropriate by SRK based on current mining and metallurgical process assumptions and certain environmental limitations. SRK has considered owner-operator for all mining and transport to processing facilities which are located approximately 20 km via sealed road from the deposit.
The angles determined for the purposes of pit optimisation and conceptual pit design are shown in Table ES 2.

Table ES 2: Open pit slope angles determined by SRK

Overall pit slope angle	Degrees
Footwall	45
Hangingwall	50

SRK used the Whittle 4X pit optimisation software to determine the economic pit limits initially for the Measured and Indicated Resources only and then incorporating the Inferred Resources to understand the upside potential. On the basis of selected optimised pit shell, SRK developed an open pit mine design using a ramp gradient of 10% which is suitable for the operation of mining trucks. SRK used a standard ramp width of 23 m dropping to 15 m for the final bench.

Figure ES 1 presents a plan view and oblique view of the design produced by SRK for the Kopsa open pit, whilst Figure ES 2 shows the site layout and the waste rock dump options. The conceptual Kopsa pit design is approximately 0.7 km long and 0.2 km wide, reaching a maximum depth of 115 m from the surface. Conceptual location of mine site infrastructure is presented in Figure ES 2.

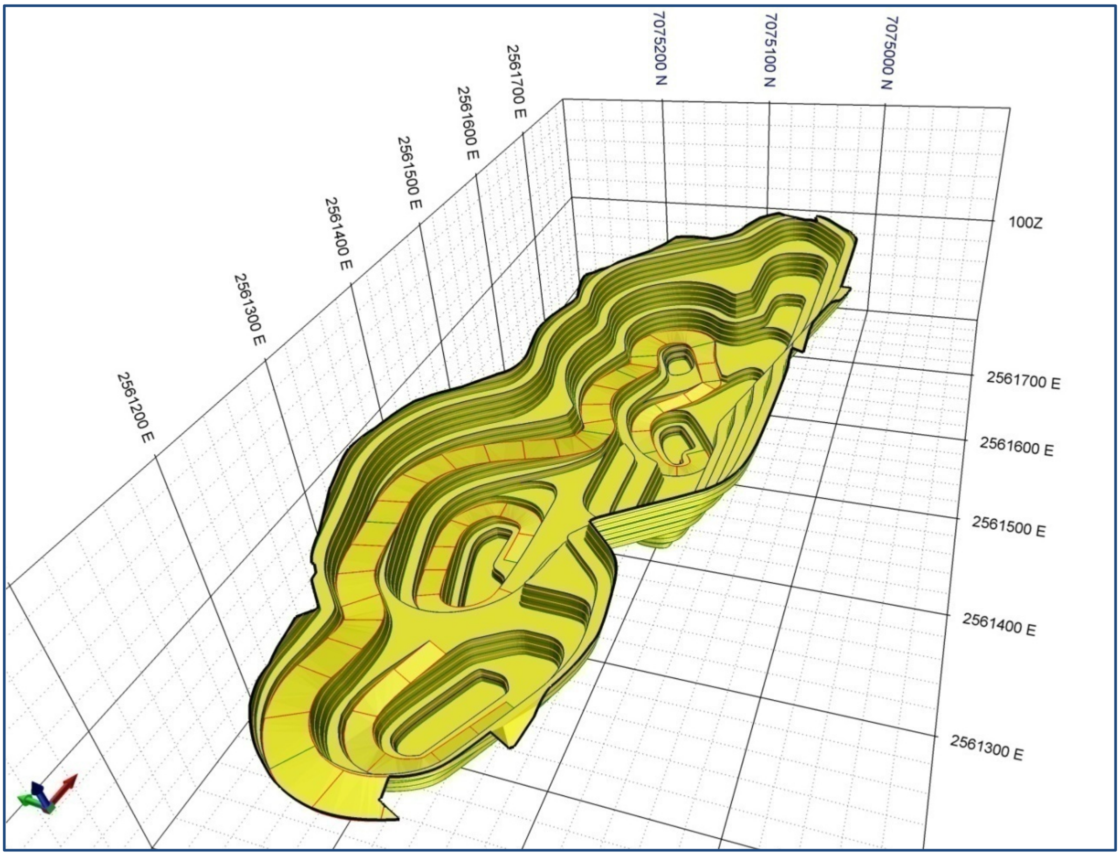

Figure ES 1: Preliminary pit design – Kopsa oblique view (Source: SRK, 2013)

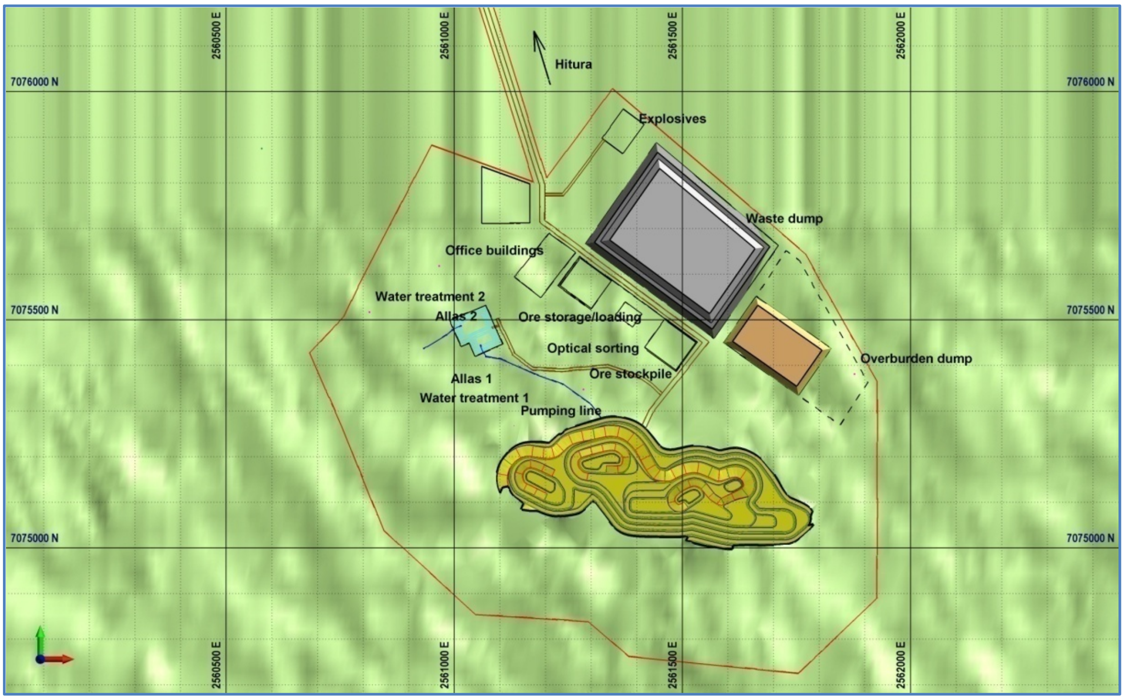

Figure ES 2: Preliminary site layout – Kopsa plan view (Source: Modified from belvedere 2013)

SRK provided a number of schedules with different mining rates to determine the optimum scenario with and without sorting processing option. Mine schedules for 0.5, 0.75, 1.0 and 1.2 Mtpa were produced. The mine plan is based on a production rate of 1.2 Mtpa which generates the highest project NPV and best mining scenario, with an overall mine life of 8 years. SRK considered a mining sequence based on three push-backs, each containing some 1.6 to 3.7 Mt of mineralised material or 2 to 5 years life. The basic mining schedule was constrained to a maximum of 6 benches (30 m) per year and there were typically 1 or 2 cut-backs being developed at any one time.

The result for mining schedule is shown in Figure ES 3.

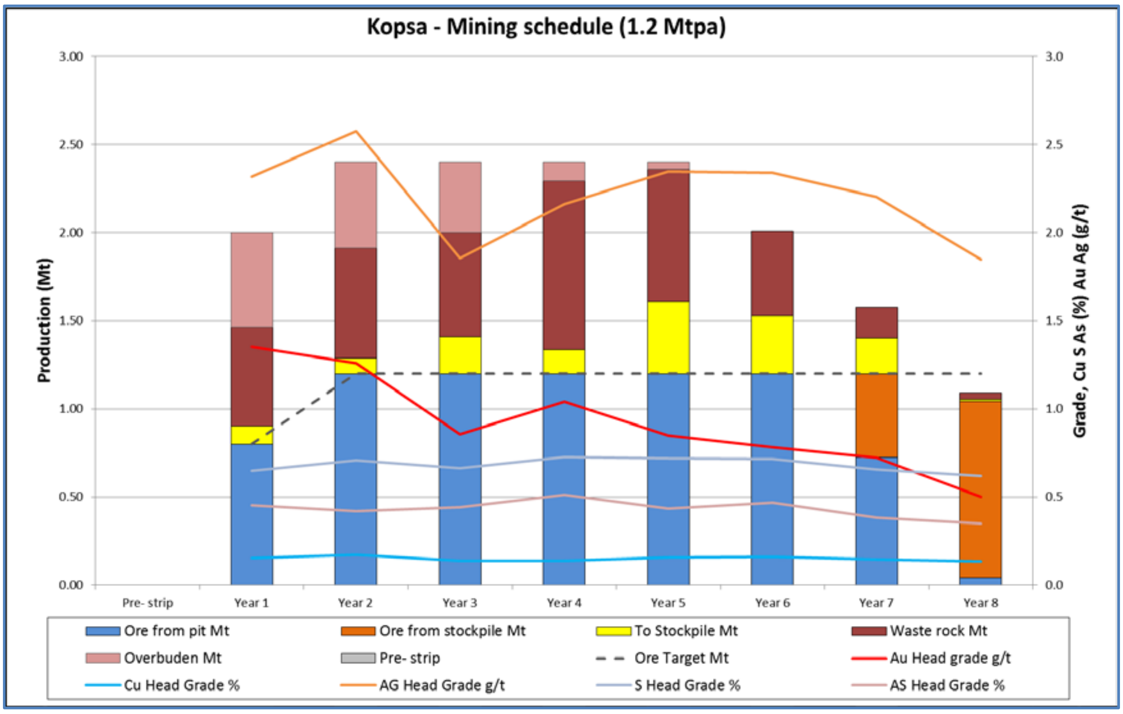

Figure ES 3: Production schedule (Source: SRK, 2013)

Equipment requirements have been determined using the following methods:

- 261 workings days per year and 16 working hours per day;
- truck and excavator requirements were calculated based on productivities and cycle times;
- 3 m^3 capacity excavators and 24 t articulated trucks have been assumed for rock mass movement
- drilling requirements have been based on 5 m benches with 115 mm diameter blasthole drills for the mineralised material and 10 m benches with 152 mm diameter blasthole drills for the waste;
- ancillary equipment has been based on material movement and primary fleet requirements;
- it has been assumed that the mineralised material from Kopsa pit will transported to the processing facility by the use of a 6 m^3 wheel loader and 40 t on-road trucks

The mine equipment requirements and productivity measured in tonnes per hour and the impact on truck requirements are shown in Figure ES 4.

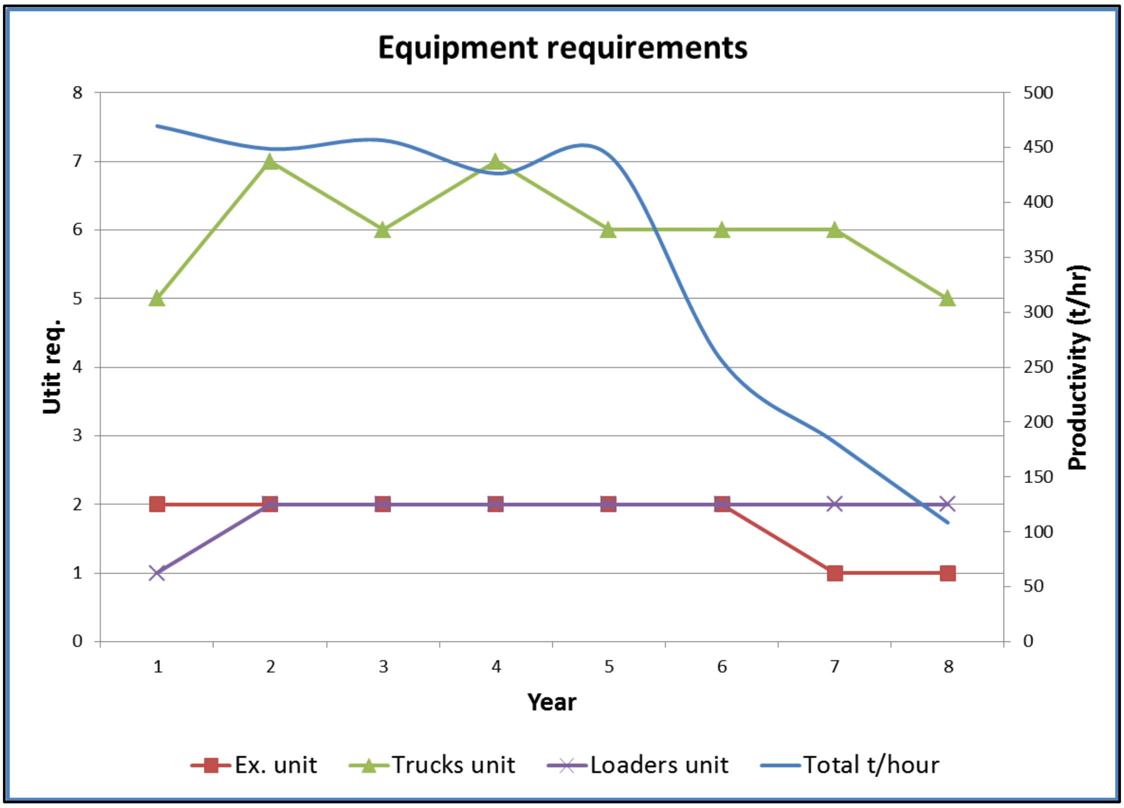

Figure ES 4: Equipment requirements (Source: SRK, 2013)

1.5 Recovery Methods

Belvedere intends to process the Kopsa material through their Hitura flotation mill, located approximately 19 km from the Kopsa site. The Hitura mill until recently processed nickel sulphide ore, at a nominal annual throughput rate of 600 Ktpa.

The existing circuit consists of a two stage crushing circuit feeding a three stage milling circuit (rod mill, ball mill, ball mill) ahead of flotation. When treating nickel sulphide ore, the flotation circuit has been configured to produce either one or two concentrate products.

In order to treat Kopsa material, the flotation circuit would be configured to produce two sulphide concentrates, a marketable copper concentrate, containing some (~40%) gold and silver, and the bulk sulphide concentrate for further processing on site. The aim of the flowsheet would be to produce a flotation tailings essentially devoid of arsenic, such that it can be stored in the existing Hitura TMF.

The bulk sulphide concentrate would be cyanide leached for the recovery of gold and silver. As indicated by the testwork, the concentrate would be reground ahead of cyanidation. Cyanidation would be followed by a conventional Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL) format, producing a smelted gold/silver doré. The tailings from cyanidation would be subjected to cyanide detoxification ahead of storage in a dedicated facility.

Sorting is being considered as part of a mine site facility that conceptually would reduce the amount of material to be trucked between the Kopsa mine site and Hitura plant site.

1.6 Tailings and Waste Management

Conventional tailings slurry requiring retention inside a paddock style impoundment was selected for the purpose of this PEA, given the experience with such tailings by the existing operations and the facilities already in place. A new tailings facility to the south of the existing facility has been assumed, with an impervious synthetic liner as a base. There is an opportunity to obtain project funding through the EU LIFE initiative. This potential funding calls for materials that is not necessarily from conventional sources, and/or may require additional handling due to the type of the material used. Exact information for achieving the EU LIFE funding is currently not fully known. It is understood that approximately M€ 5 of funding would be available.

The waste rock and overburden dumps were designed as part of the pit design. The mining activities will generate about 4.2 Mt of waste rock and the output from sorting process will generate an additional 5.8 Mt of waste that will be also disposed at the waste rock dump.

Preliminary geochemical work on the waste rock material indicate that it will be acid generating, thus requiring the collection of the water coming in contact with the waste rock material. This will require the installation of an impervious liner at the base of the dump and all water originating from the waste rock dump will be collected and directed to the water treatment plant.

1.7 Project Infrastructure

The area around the Kopsa mine site is well serviced in terms of infrastructure such as water and power, in support of the local communities and farms. As the requirements of the mine site itself will be relatively minor, the mine operations infrastructure requirements should be able to be met from the existing infrastructure in the area.

The sorting option will require additional infrastructure, particularly power, at the mine site. While there should be sufficient power transmission capacity in the vicinity of the mine to support the crushing and sorting operation, a suitable fallback position would be to generate power on site, either using stand-alone generators, or through the use of “self-contained” process units, i.e. units that have their own power source.

Given that Hitura is an existing plant site and given that the proposed throughput for the Kopsa operation is of the same order as the historic production rate for the Hitura plant, the infrastructure requirements for the Hitura plant site will be similar to those required when the plant was previously operating on Hitura nickel ore.

The most significant infrastructure requirement for the Kopsa operation will be to support the proposed haulage of material from the Kopsa mine to the Hitura plant (19 km). The mineralised material will likely be hauled using 40 t road haulage trucks and at peak production (years 2-6), SRK estimates that 7 hourly trips (or 3.5 return trips) will be required (for the base case, Scenario 6), which at this stage the Company anticipates would be the maximum permissible by the authorities, given the permanent dwellings along the proposed transport routes. Alternative routes should be considered as part of the next phase of study.

1.8 Hydrogeology & Geochemistry

During mining of the Kopsa pit, it is expected that the bulk of flow will come from the coarse grained glacial cover, in particular from the Lepola aquifer. The behaviour of surface and groundwater regimes at the Kopsa site after closure is expected to broadly reflect conditions as they existed before mining began. However, the flows local to the pit and mine complex are likely to be influenced both by the flooded pit and possibly by the partial removal or covering of sediments in the Lepola aquifer, an aquifer that given its proximity to the future project site is likely to be affected by the dewatering required to operate the mine.

The hydrogeological characteristics and distribution of geological structures in the Kopsa area is not currently understood and this will also need to be addressed by further investigation of geotechnical and exploration holes.

The Hitura tailings management facility (TMF) is expected to be expanded with new storage cells, which will modify the amount of discharge to the Kalajoki River and slightly alter the existing groundwater regime. Existing models will however need to be updated in order to re-assess remediation programmes and surrounding drawdown to the groundwater table.

Reported geology and mineralogy suggests that arsenic (from arsenopyrite) and copper (from chalcopyrite) may be mobilised through contact with the pit walls, tailings and waste rock. The quality from mine water discharges will therefore require further evaluation during the feasibility study to determine the extent of water treatment required.

1.9 Environmental and Social Assessment, Permitting and Management

The Project has a complex and lengthy approval process ahead with some uncertainty on when the environmental permit will be issued. Early robust impact evaluation is critical to reduce the risk of authorities and the public discrediting the study and delaying authorisation.

The main environmental issues relate to water management. The nature of mineralization targeted contains sulphides with elevated levels of arsenic. It appears that there is an excess of water inflow to the pit and it will be necessary to continue discharging water and tailings facility. Authorities could impose stricter limitations on the quality of water discharged to protect receiving environments including the Kalajoki River and groundwater. Water containment and treatment facilities maybe required and these have been accounted for in both operating and capital costs assumptions in the PEA economic model.

Less economically favourable transport routes may have to be considered to mitigate disturbance and risks to local communities. These various alternatives will be investigated further as part of the next phase of study.

1.10 Capital and Operating Costs

In most cases, costs were estimated by SRK for the purposes of this PEA, except in the case of operating costs for the Hitura process plant and tailings facility, which have been based to a large extent on actual operating costs from 2012.

Whilst the designed pits have been scheduled at four different production rates (500 Ktpa, 750 Ktpa, 1.0 Mtpa and 1.2 Mtpa), considering two different processing scenarios, as presented in Table ES 3 below, only capital and operating costs for Scenario 6 (production rate of 1.2 Mtpa with sorting) is discussed in this section.

Table ES 3: Production rates and processing scenarios considered as part of this PEA

Scenario	Production Rate (Mtpa)	Sorting
1	0.5	Without sorting
2	0.75	Without sorting
3	1.0	Without sorting
4	1.0	Sorting
5	1.2	Without sorting
6 = (base case)	1.2	Sorting

An overview of operating costs for the major costs centres are presented in Table ES 4 and illustrated in Figure ES 5 over the Project life of mine.

Table ES 4: Overview of operating costs by major cost centre

	USD/t moved	USD/t milled	Percentage of total
Mining	5.9	27.4	53%
Processing	3.4	15.9	30%
Tailings	0.6	2.8	5%
Environmental & Closure	0.2	1.1	2%
G&A*	0.5	2.3	4%
Contingency	0.5	2.5	5%
Total	**11.1**	**52.0**	**100%**

**G&A based on 2012 actual costs.*

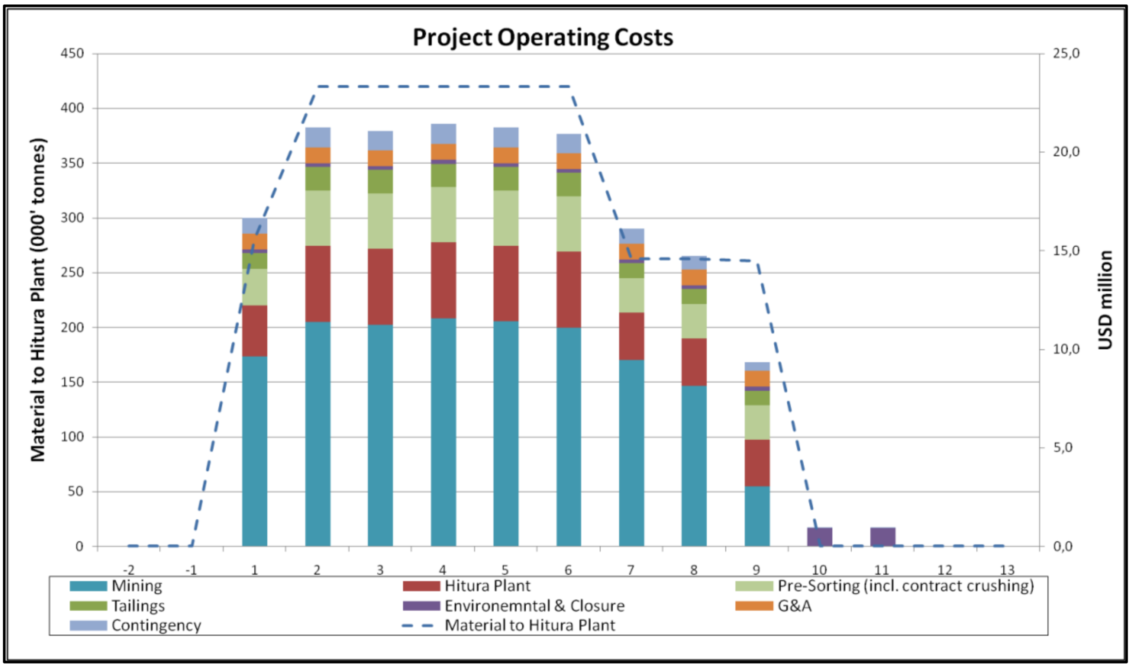

Figure ES 5: Summary of operating costs over the life of mine

The capital costs estimated as part of this study have been derived mostly by SRK, which total M USD 48 over the life of the project. SRK notes the following:

- Owner operated mining has been assumed;
- Contingencies of 25% have been applied to all capital costs;

- Working capital has been assumed at 20% of first production year operating costs;
- No provision has been made for sustaining capital, which for the purposes of this study is accounted for in operating cost provisions.
- In general (with the exception of tailings construction), capital costs have been profiled with 70% of expenditure occurring in the first pre-production year and the remaining 30% occurring in the first year of production.

Figure ES 6 gives a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres, including contingency and working capital.

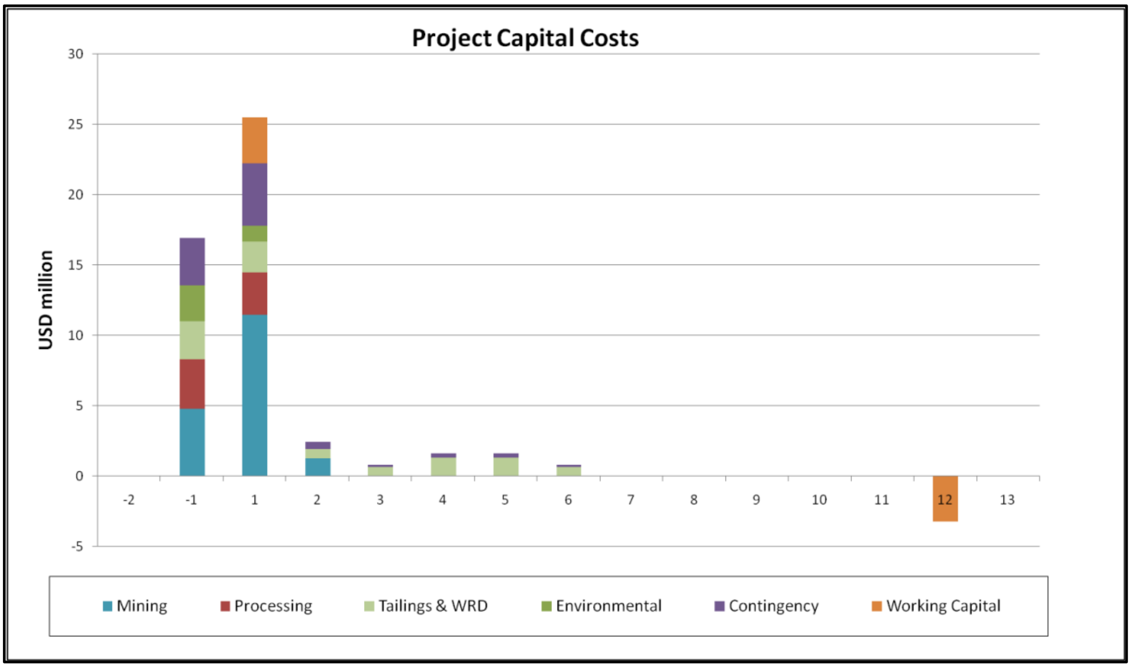

Figure ES 6: Capital cost breakdown over the LOM

Table ES 5 below presents capital cost assumptions, with a high-level breakdown under the major costs centres. Roughly 90% of capital is assumed to be required in the first pre-production year and subsequently the first two years of production.

Table ES 5: Capital cost assumptions

Description	Value (USD million)
Mining	
Mine Facilities & Haulage Dispatch System	6.1
Haul Roads	0.7
Mobile Mining Equipment	9.0
Auxiliary Equipment	2.1
Total Mining	**17.9**
Processing	
Sorting units & construction	2.2
CIL plant & refurbishments to Hitura mills	5.0
Total Processing	**7.2**
Tailings & WRD	
SRK estimate tailings construction costs	13.1
Reduction through EU Life Project funding	-6.6
Tailings back-fill plant for high sulphide material	0.3
WRD Construction (incl. ground prep & liner)	2.6
Total Tailings & WRD	**9.5**
Environmental	
Water Management Facilities (Hitura & Kopsa)	1.0
Water Treatment Plants (Hitura & Kopsa)	2.7
Land purchase (Kopsa & Hitura)	0.4
Total Environment	**4.1**
Contingency (25%)	9.7
Total	**48.3**

1.11 Economic Analysis

SRK has constructed a technical economic model (TEM) to derive a post-tax Net Present Value (NPV) for the Kopsa Project. The economic analysis contained in this report whilst including Measured and Indicated Resources only, is still preliminary in nature. Conversion of these Measured and Indicated Mineral Resources to Mineral Reserves would require the support of a pre-feasibility level study. There is no certainty that the reserves development, production, and economic forecasts on which this Preliminary Assessment is based will be realised.

The model is based on production from a single open pit mine at the Kopsa site, with on-site crushing and possible sorting based on X-ray transmission (XRT) technology. The model assumes material is trucked to the Company's existing processing facility at Hitura for production of a marketable copper sulphide concentrate and smelted gold/silver doré through conventional flotation, cyanide leaching and Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL).

For the purposes of this report, only an economic analysis of scenario 6 base case (ROM production rate of 1.2Mtpa with sorting) is discussed.

As part of the NI 43-101 process, SRK has constructed a post-tax and pre-finance TEM and assumes:

- a US Dollar (USD) valuation currency, with any Euro (EUR) derived costs being converted at a EUR:USD exchange rate of 1:0.75;

- a base case discount rate of 8%;
- the TEM is in real 2013 terms and no nominal model is presented;
- due to the uncertainty of when this project may be brought into production, the start of mining is assumed to be from ‘Year 1’ with two pre-production years (‘Year -1’ and ‘Year -2’) for the set up of basic mine infrastructure and access;
- discounting of cashflows starts in year -1;
- working capital based on 25% of the operating costs from the first year of production;
- depreciation on a declining balance basis at a rate of 20%; and
- corporate tax rate of 24.5%.

The following commodity price assumptions have been used:

- Copper USD 6 000 / tonne
- Gold USD 1 200 / troy ounce
- Silver USD 20 / troy ounce

The TEM considers the revenue and cost implications of both a marketable copper sulphide concentrate and smelted gold/silver doré.

A summary of the combined mass movement of material is presented in Table ES 6 below.

Table ES 6: Summary of movement of material from the Kopsa open pit

Mining	Unit	Value
ROM	(tonnes ‘000)	7 565
Marginal Material	(tonnes ‘000)	1 479
Waste Rock	(tonnes ‘000)	4 175
Glacial Ovb	(tonnes ‘000)	1 567
Total Material Mined	(tonnes ‘000)	14 787
Strip ratio	(Waste:Ore)	0.63
Life of mine	(years)	9
Grade Cu	(%)	0.15%
Grade Au	(g/t)	0.91
Grade Ag	(g/t)	2.21

Process recovery and concentrate grade assumptions are presented in Table ES 7 for the base case only. Smelting and Refining assumptions are presented in Table ES 8.

Table ES 7: Base case recovery and concentrate grade assumptions

Item		Unit	Base Case, Scenario 6 (ROM production rate of 1.2Mtpa with sorting)
RoM Production		tpa	1 200 000
Material Delivery to Plant		tpa	420 000
Sorting Loss	Cu	%	25
	Au	%	10
Flotation Feed Grade	Cu	%	0.32
	Au	g/t	2.34
Copper Concentrate		tpa	4 800
	Cu Rec	%	80
	Au Rec	%	40
	Cu	%	22.5
	Au	g/t	82
Sulphide Concentrate		tpa	12 600
	Au Rec	%	44.75
	Au	g/t	35.0
Cyanidation Recovery	Au	%	95
Recovery to Doré	Au	%	42.5
Overall Recovery	Cu	%	60
	Au	%	76.30

Table ES 8: Smelting and Refining assumptions

Item	Unit	Value
Copper Concentrate Losses & Deductions		
Cu Payable	(%)	95.0
Cu unit deduction	(%)	1.0
Au unit deduction	(g/t)	1
Ag unit deduction	(g/t)	30
Leach Doré		
Au Payable	(%)	99.5
Ag Payable	(%)	98.0

A summary of the results of the cash flow modelling and valuation are presented in Table ES 10 and Figure ES 7. A summary annual cashflow is presented in Table ES 9. Unit total operating costs in equivalent gold ounces is calculated by dividing total gross revenue by gold price (USD 1200/oz), divided by total operating cost.

Table ES 9: Summary Annual Cash Flow

SE443-U5522 Kopsa PEA Model															
Summary Annual Cashflow	**Units**	**Total**	-2	-1	1	2	3	4	5	6	7	8	9	10	11
ROM 1.2 Mtpa, Pre-sorting															
CASHFLOW															
Mining															
ROM	**(000' tonnes)**	**7 565**	0	0	800	1 200	1 200	1 200	1 200	1 200	725	40	0	0	0
To Stockpile	**(000' tonnes)**	**1 479**	0	0	101	87	208	135	409	328	200	9	0	0	0
Waste Rock	**(000' tonnes)**	**4 175**	0	0	562	624	592	961	753	472	174	38	0	0	0
Glacial Ovb	**(000' tonnes)**	**1 567**	0	0	537	489	400	104	38	0	0	0	0	0	0
From Stockpile	**(000' tonnes)**	**1 479**	0	0	0	0	0	0	0	0	25	710	744	0	0
Total Material Mined	**(000' tonnes)**	**14 787**	0	0	2 000	2 400	2 400	2 400	2 400	2 000	1 100	87	0	0	0
Stripping Ratio (waste / ROM)	**(w:o)**	**0,63**	0,00	0,00	1,22	0,86	0,70	0,80	0,49	0,31	0,19	0,78	0,00	0,00	0,00
Processing															
Material to Hitura Plant	**(000' tonnes)**	**3 166**	0	0	280	420	420	420	420	420	263	263	261	0	0
Au Head Grade (ppm)	**(grams)**	**2,34**	0,00	0,00	3,48	3,24	2,20	2,68	2,18	2,02	2,22	1,29	1,27	0,00	0,00
Cu Head Grade (%)	**(tonnes)**	**0,32**	0,00	0,00	0,33	0,37	0,30	0,30	0,34	0,35	0,33	0,29	0,29	0,00	0,00
Copper Concentrate Product	**(tonnes)**	**36 445**	0	0	3 265	5 526	4 467	4 484	5 087	5 170	3 054	2 711	2 683	0	0
Dore - Au	**(oz)**	**100 084**	0	0	13 170	18 415	12 524	15 209	12 388	11 447	7 879	4 582	4 470	0	0
Dore - Ag	**(oz)**	**86 974**	0	0	8 071	13 459	9 684	11 289	12 259	12 223	7 975	6 065	5 948	0	0
Revenue															
Gross Revenue															
Copper Con	**(M USD)**	160	0	0	19	28	20	23	21	20	13	9	8	0	0
Dore	**(M USD)**	122	0	0	16	22	15	18	15	14	10	6	5	0	0
Total	**(M USD)**	282	0	0	35	50	35	42	36	34	23	14	14	0	0
Net Revenue															
Copper Con	**(M USD)**	157	0	0	19	27	19	23	20	19	13	8	8	0	0
Dore	**(M USD)**	122	0	0	16	22	15	18	15	14	10	6	5	0	0
Total	**(M USD)**	278	0	0	35	50	35	41	35	33	22	14	14	0	0
Operating Costs															
Mining	**(M USD)**	**86,9**	0,0	0,0	9,6	11,4	11,2	11,6	11,4	11,1	9,4	8,1	3,0	0,0	0,0
Processing	**(M USD)**	**50,2**	0,0	0,0	4,4	6,7	6,7	6,7	6,7	6,7	4,2	4,2	4,1	0,0	0,0
Tailings	**(M USD)**	**8,9**	0,0	0,0	0,8	1,2	1,2	1,2	1,2	1,2	0,7	0,7	0,7	0,0	0,0
Environemntal & Closure	**(M USD)**	**3,5**	0,0	0,0	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,9	0,9
G&A	**(M USD)**	**7,3**	0,0	0,0	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,0	0,0
Contingency	**(M USD)**	**7,8**	0,0	0,0	0,8	1,0	1,0	1,0	1,0	1,0	0,8	0,7	0,4	0,0	0,0
Total Operating Costs	**(M USD)**	**164,6**	0,0	0,0	16,7	21,2	21,1	21,4	21,2	20,9	16,1	14,7	9,3	1,0	1,0
Unit Operating Costs	**(USD / oz AuEq)**	**700**	0	0	569	506	720	618	714	747	859	1239	801	0	0
Capital Costs															
Mining	**(M USD)**	17,9	0,0	4,8	11,8	1,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Processing	**(M USD)**	7,2	0,0	3,5	3,0	0,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Tailings & WRD	**(M USD)**	9,5	0,0	2,7	2,2	0,7	0,7	1,3	1,3	0,7	0,0	0,0	0,0	0,0	0,0
Environmental	**(M USD)**	4,1	0,0	2,8	1,2	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	9,7	0,0	3,5	4,6	0,7	0,2	0,3	0,3	0,2	0,0	0,0	0,0	0,0	0,0
Working Capital	**(M USD)**	0,0	0,0	0,0	3,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-3,3
Total	**(M USD)**	48,3	0,0	17,3	22,8	3,3	0,8	1,6	1,6	0,8	0,0	0,0	0,0	0,0	0,0
Cashflow															
Net Pre-tax Cashflow	**(M USD)**	**65,5**	0,0	-17,3	-8,0	25,3	12,8	18,0	12,3	11,3	6,1	-0,7	4,4	-1,0	2,4
Cumulative Pre-tax Cashflow	**(M USD)**	**0,0**	0,0	-17,3	-25,3	0,0	12,7	30,7	43,0	54,3	60,4	59,7	64,1	63,1	65,5
Corporation tax	**(M USD)**	**-17,8**	0,0	0,0	0,0	-5,0	-2,4	-4,0	-2,7	-2,4	-1,0	0,0	-0,2	0,0	0,0
Net Post-tax Cashflow	**(M USD)**	**47,7**	0,0	-17,3	-8,0	20,3	10,3	14,0	9,6	9,0	5,1	-0,7	4,2	-1,0	2,4

Table ES 10: DCF modelling and valuation (Base Case, Scenario 6)

Description	Units	Total
Gross Revenue	(USDM)	282
Operating costs / t total material	(USD/t)	11.1
Capital costs	(USDM)	48.3
Net post-tax cashflow	(USDM)	47.0
Payback period	(years)	3.5
Pre-tax, pre-finance NPV (8%)	(USDM)	38.6
Post-tax pre-finance NPV (8%)	(USDM)	26.4
IRR (pre-tax, pre-finance)	(%)	47.6
IRR (post-tax, pre-finance)	(%)	36.5

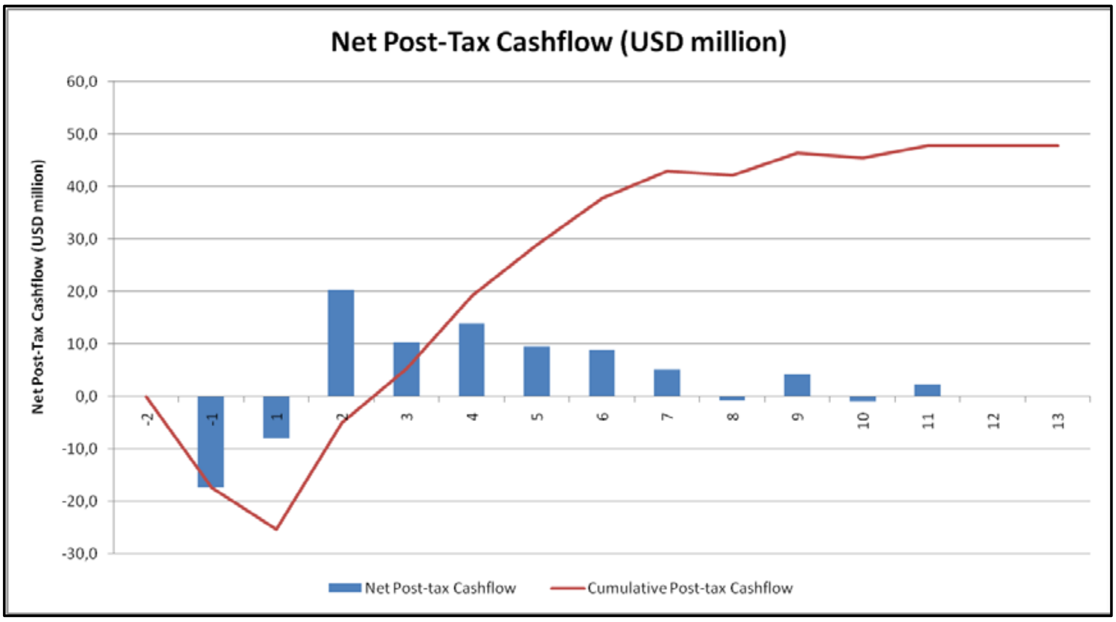

Figure ES 7: Annual and cumulative net post-tax cashflow

Figure ES 8 shows the varying NPV for varying single parameter sensitivities at an 8% discount rate for revenue, operating costs, capital costs and EUR:USD exchange rate. Table ES 11 presents the project valuation for the other production and process scenarios considered as part of this study.

Table ES 11: Summary of physical and cost assumptions for each production and process scenario, with associated post-tax valuation

Scenario		6 (base case)	5	4	3	2	1
LOM	(years)	9	9	10	10	13	19
Tonnes to Hitura plant	(Mt)	3.2	9.0	3.2	9.0	9.0	9.0
Hitura plant head grade	(Cu %)	0.32%	0.15%	0.32%	0.15%	0.15%	0.15%
Hitura plant head grade	(Au g/t)	2.34	0.91	2.34	0.91	0.91	0.91
Total Op Costs / t ROM	(USD / t)	18.2	27.0	19.1	27.9	27.1	30.1
Total Operating Costs (incl. Contingency)	(M USD)	165	244	173	253	245	273
Total Capital Costs (inc. Contingency)	(M USD)	48	70	49	69	55	54
Undiscounted cashflow	(M USD)	65.5	5.6	58.2	-1.4	19.4	-6.6
Post-tax NPV @ 8%	(M USD)	26.4	-8.0	21.8	-11.5	1.2	-11.5
Post-tax IRR	(%)	36%	-1%	31%	-5%	10%	-

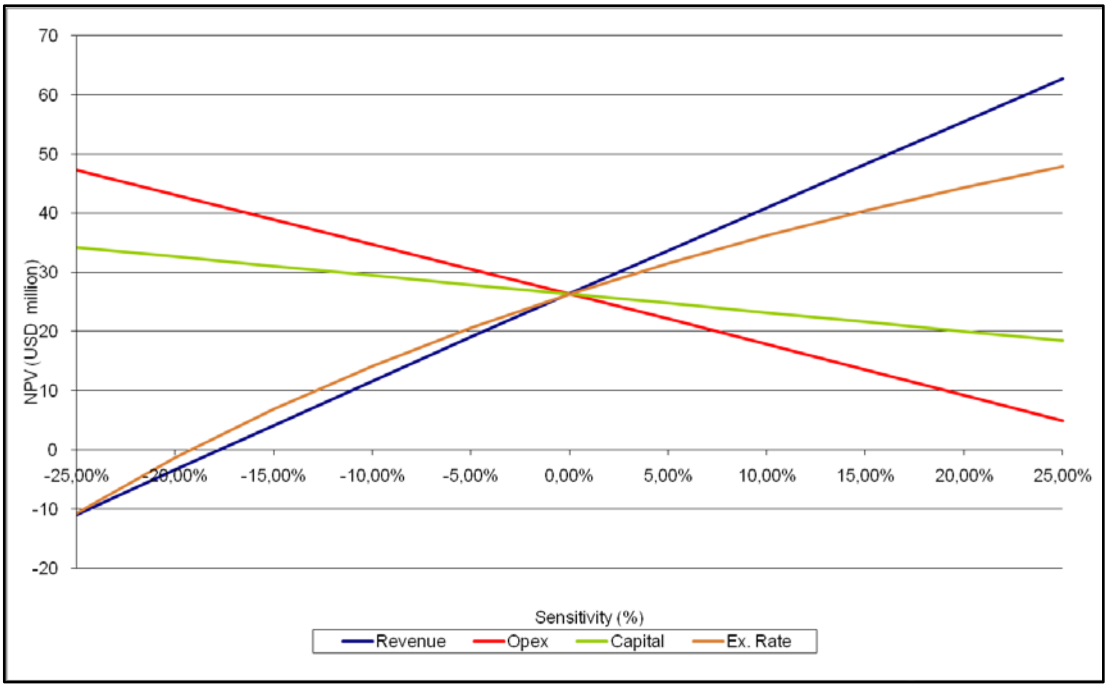

Figure ES 8: Single parameter sensitivity for base case (Sceanrio 6) post tax, pre-finance NPV at 8% discount rate.

1.12 Interpretation and Conclusions

SRK understands that the Company is proposing to undertake a feasibility study commencing in Q4 2013. SRK anticipates the work necessary to support this study will take in the order of 12 to 15 months to complete. The Company has requested that SRK provide an estimate of the costs likely to be incurred to complete the feasibility study. SRK consider that this may be in the order of USD4.5 million, including necessary drilling, ground invetigations and process testwork. A high-level breakdown of this estimate is presented in Table ES 12 below.

Table ES 12: SRK estimated costs to complete a feasibility study

Technical Discipline	USD million
Geological (incl. sterilisation drilling)	0.8
Mining	0.2
Mine Geotechnical	0.2
Hydrological	0.2
Processing and Metallurgical Testwork	2.1
Geochemistry	0.1
Tailings (incl. ground investigations)	0.3
Infrastructure	0.2
Environmental & Permitting	0.4
Total	4.5

SRK understands that the Company are involved in on-going discussions with the relevant permitting authorities. Based on these discussions, the Company anticipate having the necessary permits in place to begin production from Kopsa sometime between Q3 2015 and Q1 2016. This schedule assumes that the environmental permit application will be submitted during H2 2014, and that approval will take 12 to 18 months. The development schedule will be re-assessed during the course of the feasibility study.

1.13 Recommendations

SRK has made several recommendations regarding work that it considers should be undertaken as part of the planned feasibility study. This work is detailed later in this report. SRK is confident that should this work be included then the technical and economic viability of the Project will be properly assessed.

Most notably SRK has recommended that:

- Further developmental testwork is required for the sorting option at pilot scale. Pilot flotation testwork should be undertaken both on the product from sorting, and also on "unsorted" material.
- In addition to pilot scale testwork, laboratory scale testwork should also be undertaken on a range of samples that cover the expected variability within the deposit, in terms of head grade, mineralogy, depth and lateral extent.
- Given the relatively high volume of traffic that the project will introduce to the transport route, significant on-going stakeholder engagement will be required regarding access to this infrastructure option as the project progresses.
- Geochemical quantitative numerical predictions should be undertaken on all the waste and the pit lake that will form after closure. These predictions will aid in assessing the scale of potential impacts and confirm the suitability of selected mitigation controls. For this assessment, a full geochemical characterisation of all the materials will be required.

Certainly, in SRK's opinion, the Project justifies further work. SRK understands that the Company intends to move directly to a feasibility study and whilst there are certain risks associated with moving from a scoping level study (PEA) directly into a feasibility level study, SRK consider these risks could be mitigated by:

- Undertaking the work outlined in the recommendations section;
- Undertaking appropriate trade-off studies during the initial phases of a feasibility study;
- By the Company's intention to process material and store tailings at the Company's existing facilities at Hitura; and
- In recognition of the relatively limited size of the deposit and the Company's operating experience in the area.

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PRELIMINARY ECONOMIC ASSESSMENT FOR THE KOPSA COPPER-GOLD DEPOSIT, FINLAND

2 INTRODUCTION

The Kopsa Project is an advanced exploration project, located in Finland. It is located 8 km from the town of Haapajärvi in Northern Ostrobothnia, Finland. Belvedere Resources Ltd (Belvedere) is the parent company of Belvedere Mining Oy, which is the project owner.

This report comprises a preliminary economic assessment (PEA) of the Kopsa Project (Kopsa or the Project) and has been prepared by SRK Consulting (Sweden) AB (SRK) on behalf of Belvedere Mining Oy (the Company or Belvedere).

This Technical Report conforms to National Instrument 43-101 Standards of Disclosure for Mineral Projects (NI 43-101). SRK visited the property between 8 and 11 April, 2013.

As part of its work, SRK has prepared an independent Mineral Resource estimate for the Project and in addition, has reviewed all other technical work completed on the Project by the Company and its other contractors and consultants to a sufficient level to enable SRK to present its own opinions on the Project and to derive an audited NPV for this.

The Project is at a conceptual stage but it is currently envisaged that it will comprise a single open pit mine at the Kopsa site, with on-site crushing and possible sorting based on X-ray transmission (XRT) technology. Material will then be trucked to the Company's existing processing facility at Hitura for production of a marketable copper sulphide concentrate and smelted gold/silver doré through conventional flotation, cyanide leaching and conventional Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL). Subject to financing, the Company expects to commence a feasibility study in Q4 2013.

The work undertaken by SRK in compiling this report has been managed by Mr Johan Bradley (CGeol FGS, EurGeol) and reviewed by Dr Mike Armitage (CGeol FGS, CEng MIoM3). Both Mr Johan Bradley and Dr Armitage are Qualified Persons (QP) as defined in National Instrument 43-101 of the Canadian Securities Administrators (NI 43-101).

The details of the various contributing authors and their respective areas of technical responsibility are presented in Table 2-1 below.

Table 2-1: Contributing authors and respective area of technical responsibility

Contributing Author	Area of technical responsibility	Sections of this report
Johan Bradley (QP)	Geology and Technical Economic Model	1 to 11, 21 & 22
Lucy Roberts	Resource Estimation	12 & 14
Chris Bray	Mine Optimisation, Design and Scheduling	16
David Saiang	Geotechnical assumptions	16.2
John Willis	Process Metallurgy, Infrastructure, Markets and Concentrate Transport	13,17 and 18
Michel Noël	Tailings Dam Design and Waste Rock Dumps	20.3, 20.4
Matt Dey	Acid Rock Drainage and Metal Leaching	20.2
William Harding	Hydrology and Water Management	20.1
Richard Evans	Environmental, Permitting and Social Impacts	20.5
Dr Mike Armitage (QP)	Review	All sections

For the purposes of this report, the following persons act as QP: Johan Bradley and Dr Mike Armitage. Appropriate QP certificates for these individuals accompany this report.

2.1 Basis of Technical Report

This report is based on information collected by SRK during a site visit performed between 8 and 11 April 2013 and on additional information provided by Belvedere throughout the course of SRK's investigations. Other information was obtained from the public domain. SRK has no reason to doubt the reliability of the information provided by Belvedere. This technical report is based on the following sources of information:

- Discussions with Belvedere personnel;
- Inspection of the Kopsa Project area, including outcrop and drill core;
- Inspection of existing operating process and tailings facilities at Hitura;
- Review of exploration data collected by Belvedere; and
- Additional information from public domain sources.

2.2 Declaration

SRK's opinion contained herein and effective 02 October 2013 is based on information collected by SRK throughout the course of SRK's investigations, which in turn reflect various technical and economic conditions at the time of writing. Given the nature of the mining business, these conditions can change significantly over relatively short periods of time. Consequently, actual results may be significantly more or less favourable.

This report may include technical information that requires subsequent calculations to derive sub-totals, totals and weighted averages. Such calculations inherently involve a degree of rounding and consequently introduce a margin of error. Where these occur, SRK does not consider them to be material.

SRK is not an insider, associate or an affiliate of Belvedere, and neither SRK nor any affiliate has acted as advisor to Belvedere, its subsidiaries or its affiliates in connection with this project. The results of the technical review by SRK are not dependent on any prior agreements concerning the conclusions to be reached, nor are there any undisclosed understandings concerning any future business dealings.

3 RELIANCE ON OTHER EXPERTS

Sections 4 to 10 of this report are to some degree extracts from the Company's existing technical reports, in particular Kopsa Au-Cu Property, NI 43-101 Technical Report by Pym, D., Strauss, T., Merlinänen, M, Lovén (2012).

The additional information reviewed in preparing this report has also largely been provided directly by the Company and its associated consultants, contractors and business partners. SRK has conducted face to face meetings with those consultants responsible for certain technical aspects of the Projec to enable it to take responsibility for the assumptions given here.

SRK has also confirmed that the Mineral Resources reported herein are within the exploration claim boundaries given below and that the exploration and mining leases presented by the Company reflect the publicly available information. SRK has not, however, conducted any legal due diligence on the ownership of the exploration permits or exploitation concessions themselves and has relied upon the Company's legal advisors (KallioLaw), who present their opinion of the Company's legal tenure over Hitura and Kopsa in a letter dated 30th September 2013, which SRK has reviewed.

4 PROPERTY DESCRIPTION AND LOCATION

4.1 Description

The Kopsa project is an advanced Cu-Au exploration project in Haapajärvi community, in Northern Ostrobothnia, Finland (Figure 4-1). The deposit is centred around Finnish National Coordinate System (KKJ – Kartasto Koordinaatti Järjestelmä): X: 2561400; Y: 7075150, with the following latitude and longitude: N63 46.23, E25 14.20.

Figure 4-1: Location of the Kopsa project (Source: GoogleMaps. July 2013)

4.2 Property Ownership

A mining lease for Kopsa was applied for in March 2009, and is currently pending. Until the mining lease application has been processed the two exploration claims (7405/1 and 7686/1) remain valid. An extension to the mining lease application was submitted in 2010.

As commented above, SRK has confirmed that the Mineral Resources reported herein are within the exploration permit boundaries given below and that the exploration permits and exploitation concessions as presented by the Company reflect the publicly available information at the Mining Inspectorate of Finland (Tukes). SRK has not, however, conducted any legal due diligence on the ownership of the exploration permits or exploitation concessions themselves, but has rather relied upon a legal opinion as to the integrity of these provided by Kallio Law and discussed in Section 3.

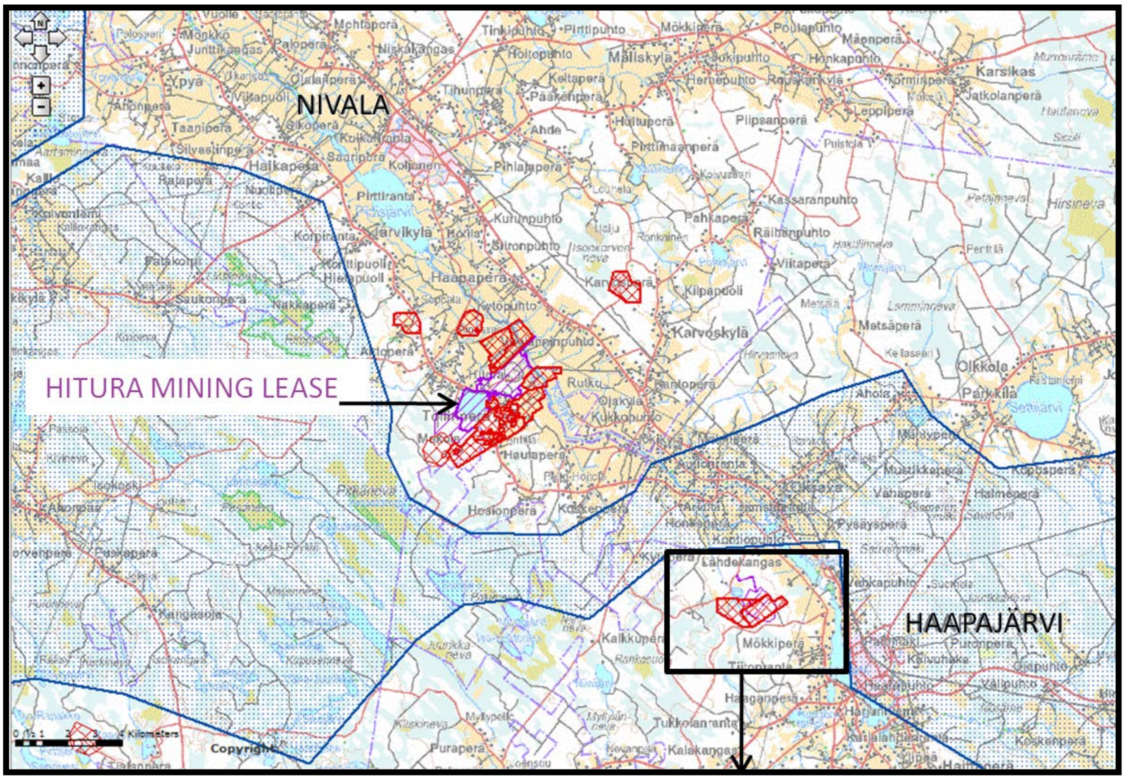


Figure 4-2: **Kopsa project marked out with black box. Approximately 20 km, to Belvedere Hitura mine (Source: GTK, July 2013).**

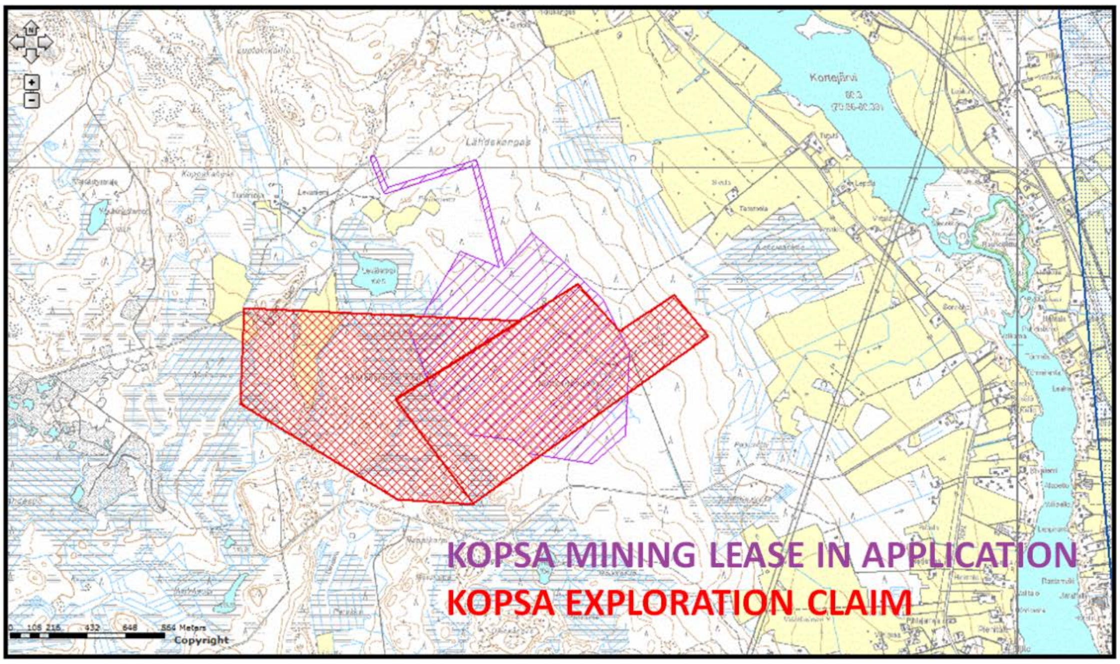


Figure 4-3: **Showing the area in the black box from previous figure. Exploration claims in red and area covering the Mining Lease applied for in purple (Source: GTK, July 2013).**

4.3 Additional Permits, Royalties and Payments

SRK is not aware of any known environmental liabilities, royalties, back-in rights, payments or other encumbrances to which the property is subject. All the payments for damage compensations are also up to date as far as SRK is aware.

4.4 Surface Rights

For the purposes of this report, all surface rights are covered by the existing exploration claims and mining leases and lease applications as detailed above. Additional permitting is required prior to commencement of mining operations and a discussion of these is presented in Section 20, below.

5 ACCESSIBILITY, CLIMATE, LOCAL RESOURCES, INFRASTRUCTURE AND PHYSIOGRAPHY

5.1 Access

The Kopsa property is located 8 km by road from the town of Haapajärvi. The property can be accessed by 2 to 3 km of forestry gravel road, leading from the sealed Tiitonranta road off the Haapajärvi-Reisjärvi Highway no. 58.

As is common with much of Finland the existing service infrastructure is excellent. The claim area is 8 km by road from the nearest railhead (at Haapajärvi) on the Nivala – Ylivieska railway line, which in turn is well connected to Oulu and/or Kokkola, two sea-port cities of Finland. The main railway line to Kokkola port (and Boliden's zinc smelter) runs through the town of Ylivieska. The nearest commercial airports are also at Kokkola (120 km by road to the west) and Oulu, (165 km by road to the north), with regular daily flights to Helsinki.

5.2 Physiography and Climate

The topography of the Kopsa property is fairly plain and covered with birch, spruce and pine forest. The other occasional species of vegetation in the area is aspen. There is only one rock outcrop which is in the central part of the deposit. The elevations of the Kopsa property vary between 100 and 120 m above mean sea level (MASL). The only water body in the vicinity of the deposit is Levälampi lake (200 x 250 m) with an elevation of 106.5 m. The Kalajoki River flows approximately 2.5 km to the East of main Kopsa deposit. The overburden thickness on the property varies from 0 m in the main Kopsa zone, where the outcrop is to approximately 25 m to the western part where small sand/ till hills are located.

Kopsa and its surrounding region belong to the temperate coniferous-mixed forest zone (Taiga/Boreal) with a climate described, according to the Koppen Climate Classification, as snow climate (D) fully humid (f) with cool summers (c). Winters (November-April) are cold and wet with an average temperature of -9°C and a range between -5 and -30°C. During the temperate summer period (June-August) the temperature ranges between 10°C and 25°C, with an average of 15°C.

The average monthly precipitation recorded by the Finnish Meterological Institute (FMI) at the nearby Haapavesi weather station ranges between 25mm in February and 75mm in July as indicated in Figure 5-1.

The estimated rainfall for a 1 in 30 year 24-hour event is approximately 60 mm. The estimated rainfall for a 1 in 100 year 24-hour rainfall event is approximately 80 mm. These rainfall totals have been calculated to support cost estimates and conceptual designs in this study. Estimates are based on baseline data obtained for the Kopsa Project and on SRK's experience previously gained from similarly sized projects in the same climate zone. It is however important to note that climate data can easily vary on a local scale and that these values are indicative only. More climatic data needs to be collected to add confidence to these preliminary estimates and should be obtained during the next stage of technical evaluation.

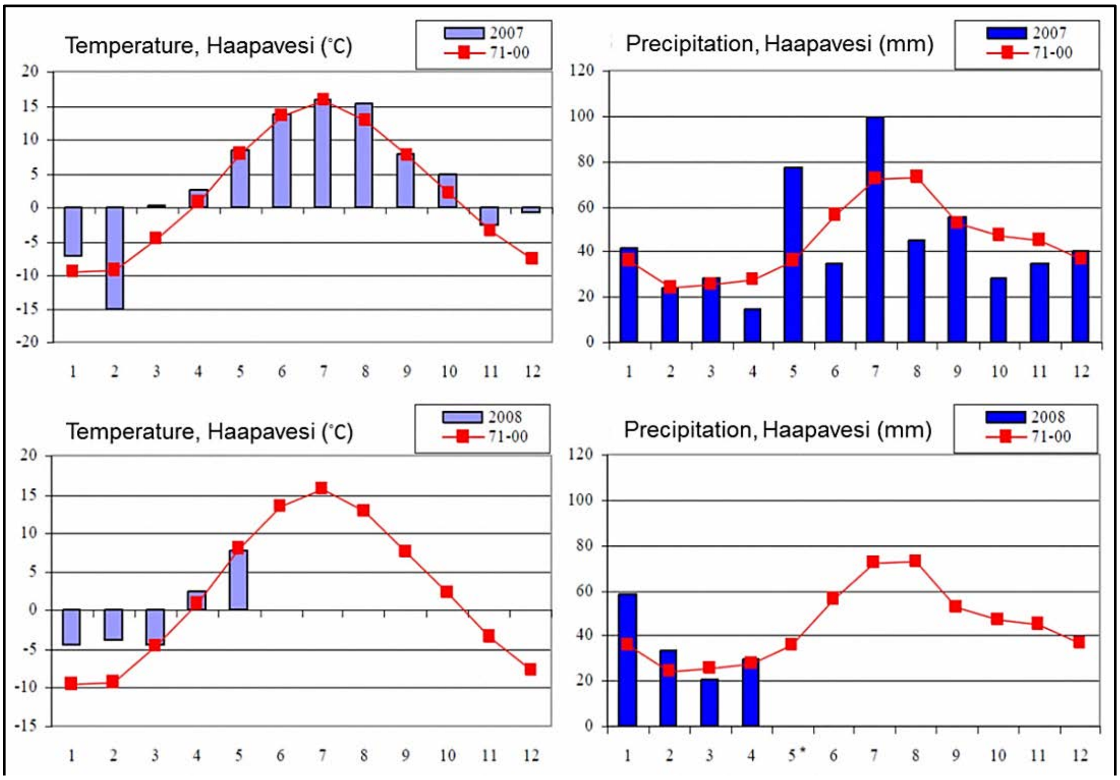


Figure 5-1: Monthly temperature and precipitation averages measured at the Haapavesi weather station (c. 40km from Kopsa) (Modified from: Baseline Monitoring report for the Kopsa Area (LVT, 01.07.2008). Source: Finnish Meteorological Institute, Climate Services, July 2013)

5.3 Local Resources and Regional Infrastructure

Logistical infrastructure is readily available in the area. A high voltage power line crosses the property to the south-east, approximately 1.0 – 1.5 km to the East of the main zone of mineralisation. Water is readily available from either the Kalajoki River (~2 km) or Levälampi Lake (~1 km). The Hitura nickel-cobalt mine and metallurgical plant (owned 100% by Belvedere) is approximately 13 km distance from the Kopsa deposit, provides a suitable site for processing of potential mined material and for potential tailings storage areas, potential waste disposal areas, and possible heap leach pad areas. It is likely that the Kopsa mineralised material will be hauled to the Hitura mine plant for treatment, a road distance of approximately 20 km. The nearest village is Tiitonranta approximately 2 – 3 km to the ESE of the deposit. There should be good availability of potential employees given that the area has a long tradition of mining.

6 HISTORY

6.1 Introduction

The Kopsa Au-Cu-Ag-As zones and associated mineralization of the Sorola occurrence has been an exploration target since 1937, when boulders were traced to the Kopsa outcrop. The area has had numerous studies undertaken on it by various exploration companies.

6.2 Ownership

The ownership of the property has changed hands several times since exploration began in 1939, as shown in Table 6-1.

Table 6-1: Historic property ownership and exploration campaigns

Name of the Company	Period of exploration
North Finland Research Foundation	1943 – 1954
Geological Survey of Finland (GTK)	1939, 1961, 1983 – 1985
Outokumpu Oy	1940 – 1941, 1964 – 1966, 1971 – 1973, 1977 – 1978, 1980 – 1982
Baltic Minerals Finland / Glenmore Highlands Inc	1995 – 1999
Belvedere Resources Finland Oy (subsidiary of Belvedere Resources Ltd)	2002 - 2009
Finn Nickel Oy (subsidiary of Belvedere Resources Ltd)	2009-2010
Belvedere Mining Oy (subsidiary of Belvedere Resources Ltd)	2010 - present

6.3 Historic Exploration

The North Finland Research Foundation (1943 - 54) and Geological Survey of Finland (1939, 1961, 1983-85) undertook surveys of glacial erratic boulders, bedrock mapping, diamond drilling, till geochemistry, IP surveys and magnetic surveys. In the 1980s the GTK exploration was focused chiefly around the Sorola satellite occurrence.

Outokumpu completed numerous works including till geochemistry, stratigraphic mapping, ground IP, a conductivity (Slingram) and magnetic survey, mineralogy study, diamond drilling in various phases (1940 - 41, 1964 - 66, 1971 - 73, 1977 - 78, 1980 - 82). Bedrock mapping, trenching, percussion and diamond drilling, VLF surveys, and a pilot flotation processing study were also completed.

Baltic Minerals/Glenmore Highlands completed a diamond, percussion and RC-drilling programme, trenching and ground magnetic survey, bedrock mapping in the surrounding areas, and a geochemical till survey between 1995 and 1999.

A number of different owners have carried out drilling campaigns on the property, with drilling commencing initially in 1939. Table 6-2 lists the previous owners, when they have been active, the type of drilling undertaken and number of holes completed.

Table 6-2: List of drilling campaigns conducted to date

Company	Period	Type	Holes	Supporting 2013 Mineral Resource estimate
Outokumpu Oy	1939-1944	Diamond	44	
GTK	1961	Diamond	3	
Outokumpu Oy	1970s	Diamond	50	
		Percussion	200	
Outokumpu Oy	1981-1983	Diamond	29	
GTK	1984-1985	Diamond	13	
		Percussion	432	
Glenmore Highlands (GMHL)	1995-1997	RC	32	
		Diamond	18	X
Total		Diamond	157	
		RC	32	
		Percussion	632	

A Historical resource estimate for the Kopsa deposit was discussed in a paper in Economic Geology (Gaál and Isohanni, 1979). In this paper, a geological resource, estimated by Outokumpu Oy is stated as 24.6 Mt grading 0.18% Cu, 0.57 ppm Au, 4 ppm Ag and 0.36% As, with historic "proven plus probable reserves" quoted at 1.1 Mt grading 0.17% Cu, 1.9 ppm Au, 4 ppm Ag and 0.81% As. No further information on how these estimates were derived is provided in the paper. SRK has also not verified these estimates.

7 GEOLOGICAL SETTING AND MINERALIZATION

7.1 Regional Geology

Geologically the Kopsa Property belongs to the Raahe-Ladoga zone (e.g. Korsman 1988, Ekdahl 1993), which runs parallel to the Archaean craton margin (Figure 7-1) and the geology of the area is the product of complex Palaeoproterozoic subduction and collision processes (Gaál 1986 and 1990). The Raahe-Ladoga deformation zone is divided into different shear zones especially in the north western part of the zone and is the most important sulphide ore zone in Finland according to the amount of deposits and occurrences.

Central Ostrobothnia consists of moderately to strongly metamorphosed, in places also intensively sheared, Palaeoproterozoic rocks (Kousa et al. 2000). The supracrustal rocks are mostly migmatised mica gneisses intercalated with minor quartz-feldspar gneisses, graphite and mica schists and amphibolites of volcanic origin and locally with some dolomite and skarn. Volcanic rocks (mainly felsic and mafic) have only limited extension, but host numerous massive sulphide deposits (Pyhäsalmi, Vihanti).

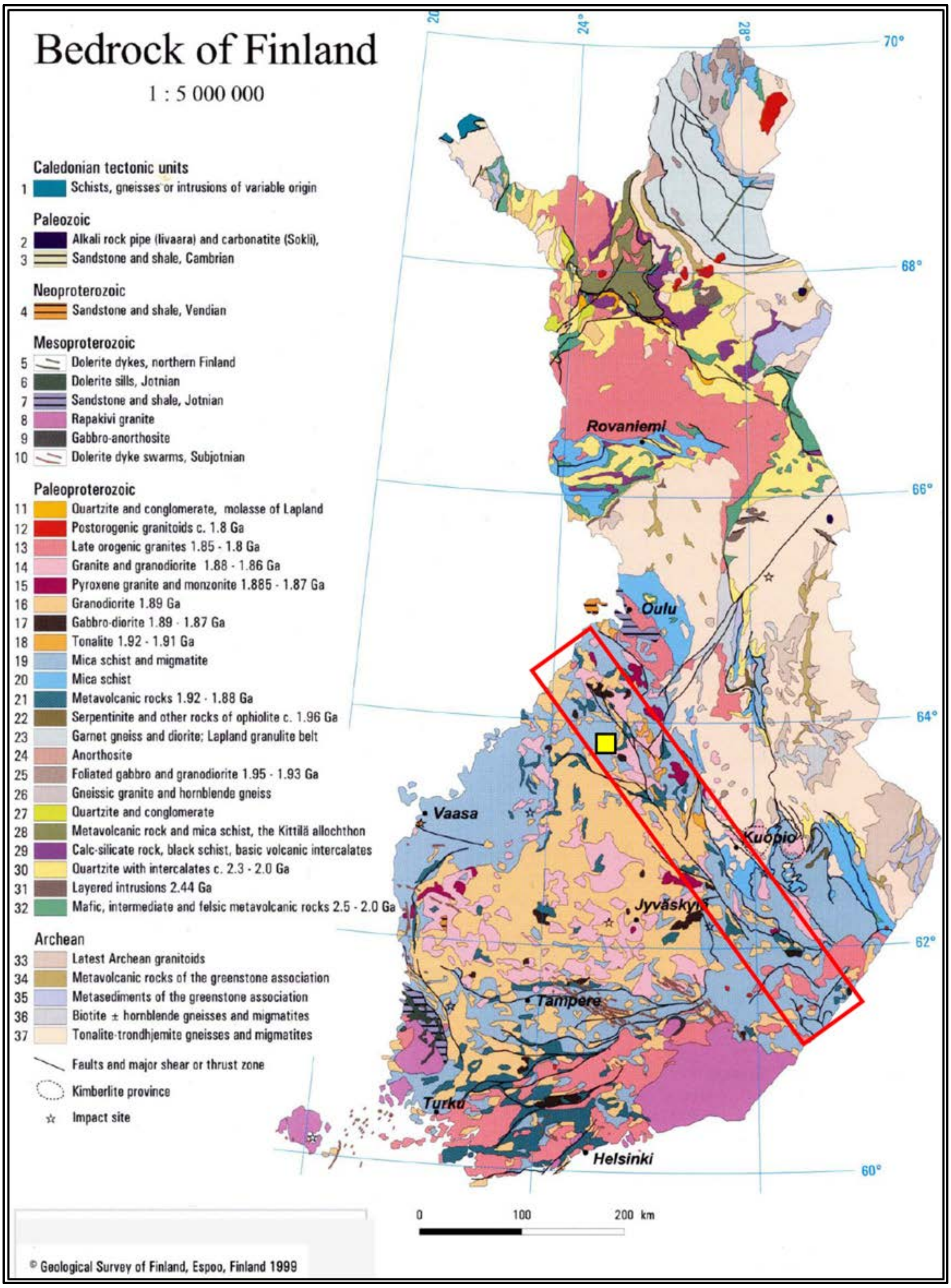


Figure 7-1: Geology of Finland. Kopsa shown as Yellow square (Source: GTK, July 2013).

7.2 Local Geology

The Kopsa Cu-Au deposit is hosted by a Proterozoic late orogenic intrusive body, known as the Kopsa Tonalite. The unit is rhomb shaped and is approximately 1,200 x 550 m in extent at surface. The unit is intruded into a package of meta-greywacke, mica schists, and intermediate pyroclastic volcanics, has been dated at approximately 1.92 Ga, and is interpreted as a turbidite sequence. All lithologies have been metamorphosed to upper-greenschist to amphibolite facies. A local geological map is shown in Figure 7-2.

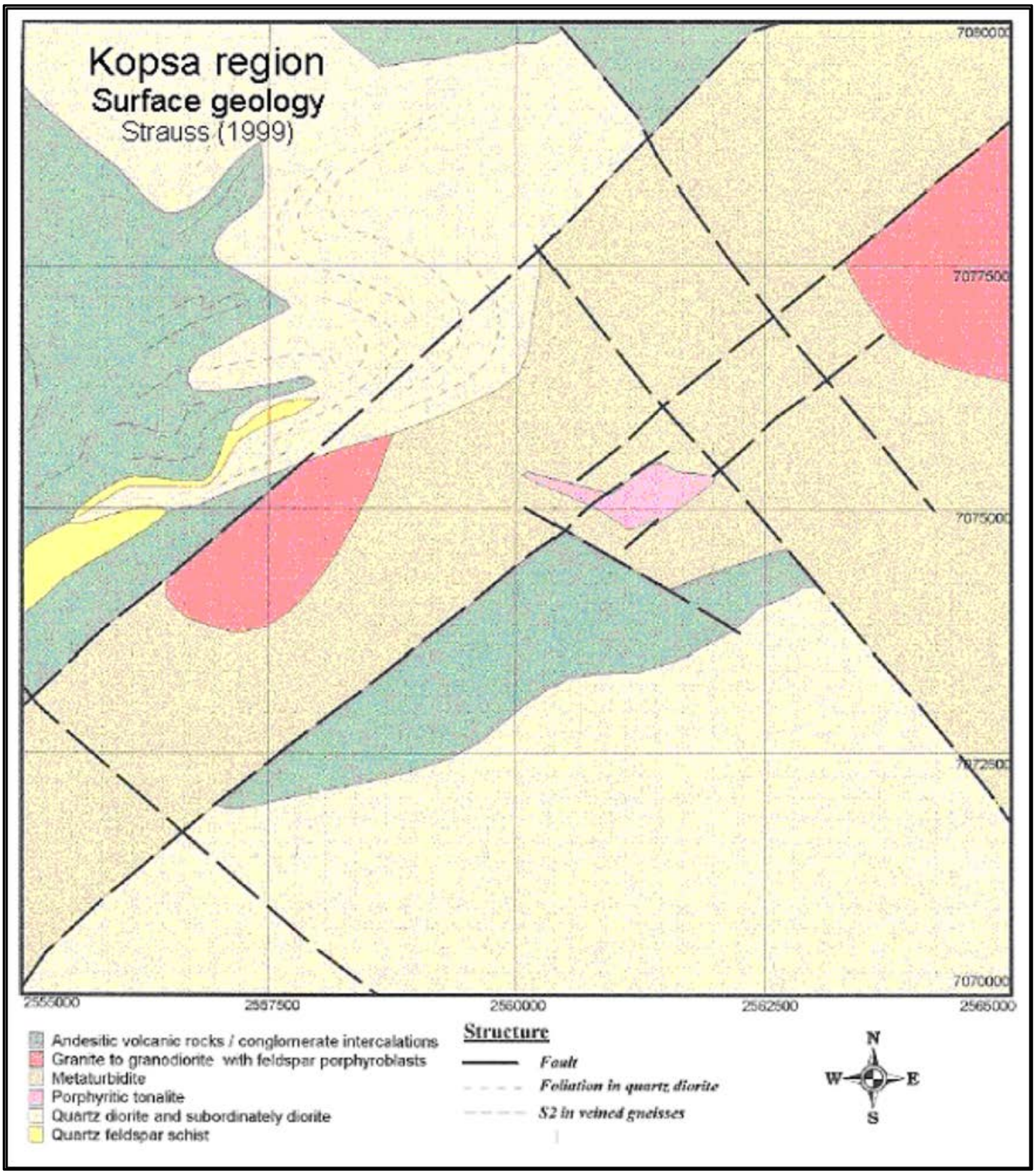


Figure 7-2: Kopsa local geology. Kopsa deposit within porphyritic tonalite (Source: Strauss (1999)).

7.3 Mineralisation

The gold mineralisation at Kopsa is associated with quartz and arsenopyrite veining. The mineralization occurs as compact sulphide veins, or as stringers and blebs in connection with quartz veining and silicification. Fine grained disseminated mineralization also occurs outside the main veins, in the altered host rock. In the higher grade areas, the quartz veins and silicification form a stockwork structure. The mineralised body extends for approximately 700 m along strike, 200 m down dip, with a maximum thickness of 50 m. The approximate strike of the mineralization is 105° (approximately east west), with a shallow dip of 20º towards the south.

The typical mineralogy of the deposit includes main sulphide minerals such as arsenopyrite, chalcopyrite, pyrrhotite, and pyrite. Gold occurs as free gold (non-refractory), and has a close association with bismuth.

8 DEPOSIT TYPE

Kopsa has been classified variably as a Proterozoic porphyry copper, subsequently overprinted by an orogenic gold system, to an orogenic gold deposit to more recently an intrusive related type of deposit. The evidence for a porphyry origin is weak and is largely based on the potassic alteration and the copper gold association. Mineralization however appears to have occurred just above the brittle ductile boundary at crustal levels > 10 km. Contacts with the country-rock are passive and appear in thermal equilibrium, further supporting a deep level of formation below that associated with porphyry coppers. Likewise the model for orogenic gold is not conclusive and is based largely on a quartz vein gold association. Alteration and sulphide associations suggest fluids with at least some magmatic input, and the host intrusion also cuts the main regional d 2 foliation associated with peak deformation suggesting mineralization post-dates orogenesis.

Belvedere has tentatively classified the deposit as belonging to the intrusive related style of deposits, based primarily on mineralogy and mode of occurrence. Based on mineralogy, the host intrusion is at best weakly oxidised and probably reduced, the base metal enrichment is high relative to gold grade factors typical of intrusive related gold systems. The bismuth, antimony, tellurium, molybdenum association and alteration suggest high temperature fluids with at least some magmatic component and there is a regional association of gold mineralization with the margins of large intrusive complexes. In SRK's opinion, until further studies are done, particularly on the composition and age of ore forming fluids relative to the intrusive host there can be no conclusive type-classification of the deposit.

9 EXPLORATION

9.1 Introduction

Belvedere Resources has conducted numerous exploration programmes at Kopsa since the property was first acquired. These include ground geophysical surveys, geochemical surveys, structural studies, percussion drilling, and six phases of diamond drilling. Previous owners also conducted numerous exploration programmes in the area. The ground geophysical survey and structural studies are most material to the Project and these are discussed briefly below.

9.2 Geophysics

The most recent geophysical survey was an induced polarisation (IP) / Resistivity survey carried out by JVX of Canada in 2003/2004, and conducted on behalf of Belvedere. This only covered the western half of the intrusion and extended out into the panhandle to the west. The results of this survey were merged with an earlier Glenmore Highlands survey carried out in 1997. The composite map with land tenure and the extents of the current modelled mineralization is shown in Figure 9-1. The Company consider IP to be a useful guide to sulphide mineralization found at the Kopsa and results from this may be used to help guide exploration drilling in the future.

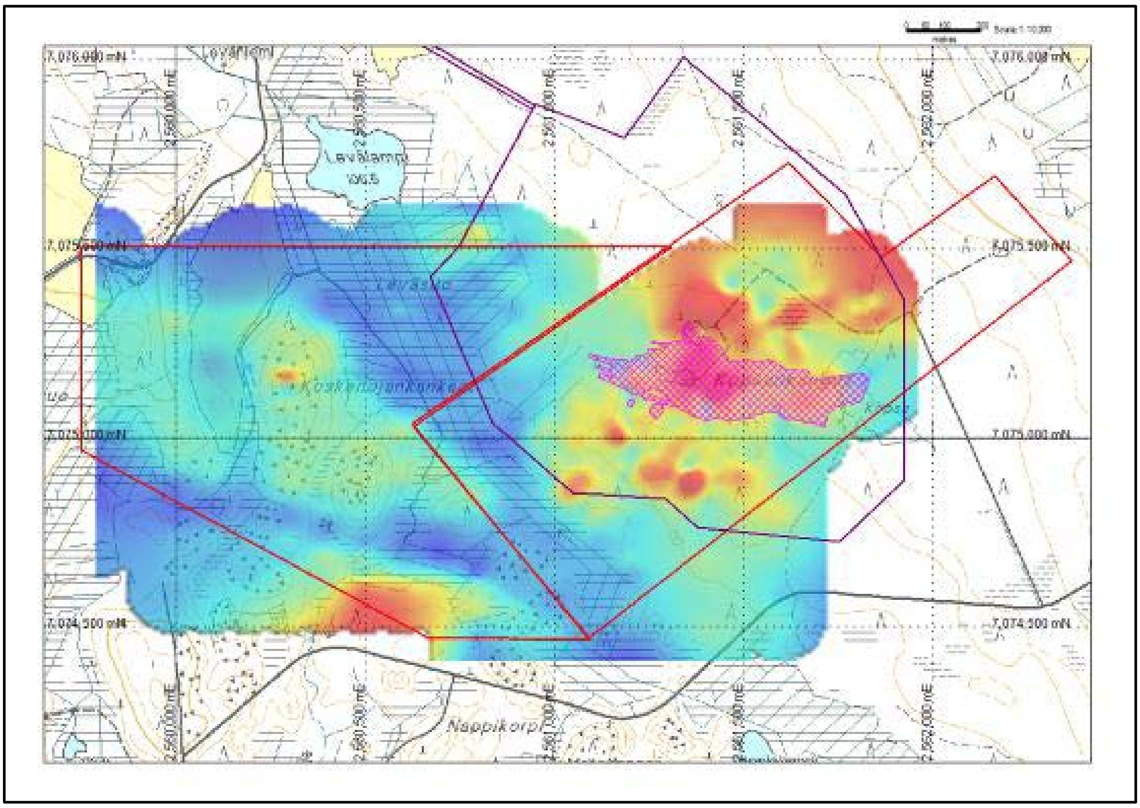


Figure 9-1: Compilation map of IP surveys with 2012 modelled mineralisation (Source: Belvedere, July 2013)

9.3 Structural Study

A macroscopic study of the structural features in the limited outcrops of the Kopsa deposit has been carried out by the Geological Survey of Finland (GTK), by Sorjonen-Ward (2005).

In total 9,952 structural measurements were recorded from 19 drillholes. The main structural features recorded are shear zones and vein sets (both with and without mineralization). The study revealed that a predominant NNW direction can be observed in the Au-rich veins analysed. These appear to be constrained within the low-grade halo, which has a shallow-dip towards the south. Varying directions were discernable in the various areas of the deposit – such as the ‘outcrop zone’, ‘north zone’ and ‘main zone’.

10 DRILLING

10.1 Summary

Table 10-1 summarises the drilling completed on the project to date and Figure 10-1 shows the resulting distribution of the drillhole collars throughout the deposit area in relation to the SRK mineralization interpretation. A typical cross-section through the Kopsa deposit is shown in Figure 10-2, with Cu and Au grades shown.

Table 10-1: Overview of Belvedere drilling campaigns

Company	Period	Type	Holes
Belvedere	2002-2007	Percussion	48
		Diamond	32
	2010	Diamond	31
	2011	Diamond	45
Total		Diamond	103
		Percussion	48

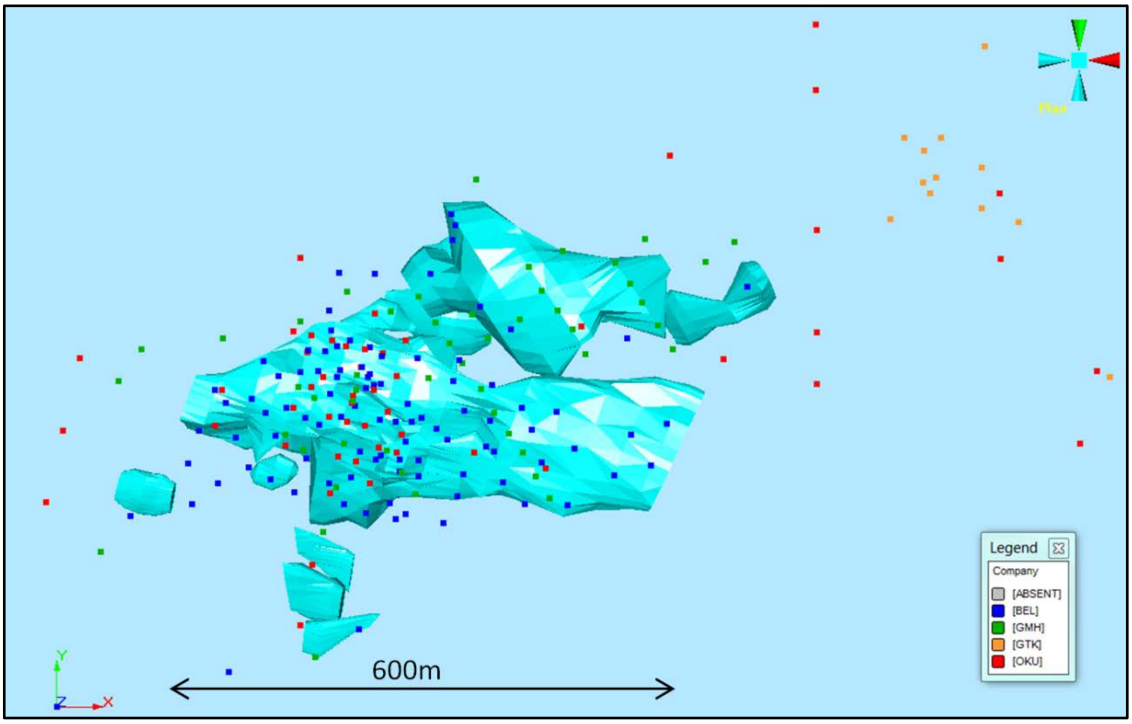


Figure 10-1: Company and historic drillhole collars coloured by company. Blue = Belvedere (BEL), Green = Glenmore Highlands (GMH), Orange = GTK, Red = Outokumpu (OKU) (Source: SRK, July 2013).

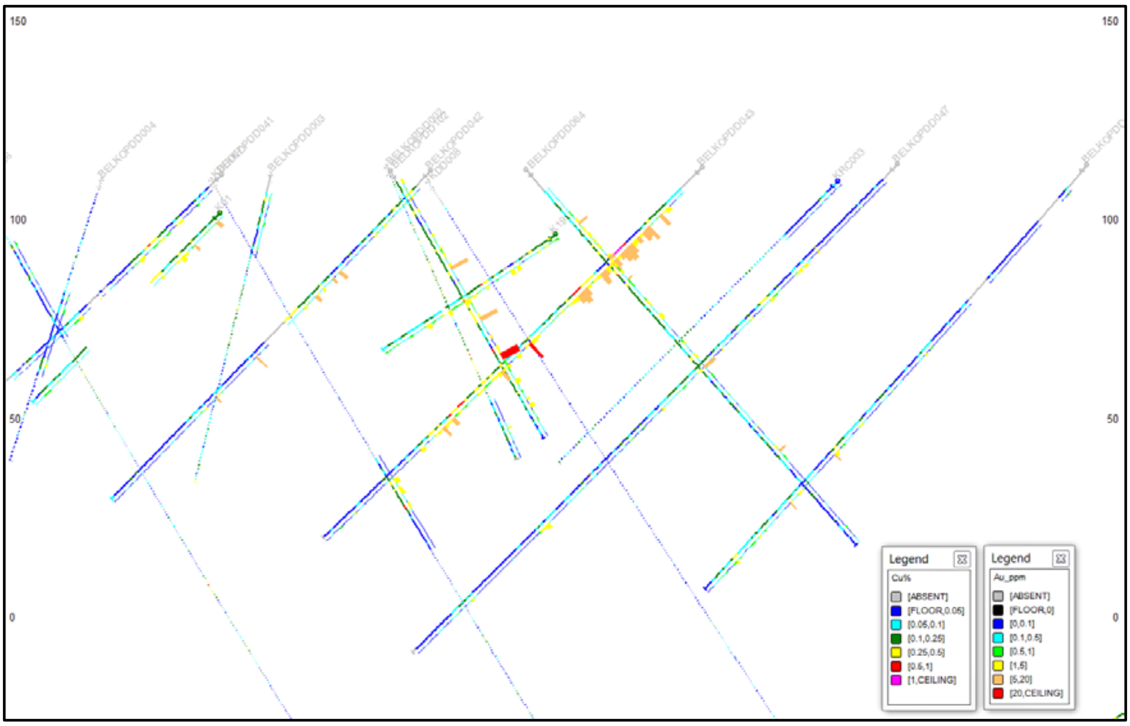


Figure 10-2: Typical cross-section (looking west) through the central area of the main mineralization at Kopsa. Drillholes coloured by Cu% (down-hole) and Au (g/t) (histogram on right) (Source: SRK, July 2013).

10.2 Down-hole Surveys

The holes drilled by Belvedere, which are marked with the prefix BELKOPDD, have been surveyed down the hole at 5 m intervals. The 2003 drilling campaign used Maxibor for surveying while those done in 2006, 2010 and 2011 used EMS, Deviflex and Gyroflex, and Reflex respectively. All surveying methods were consistent with industry best practice at the time of drilling. As the host lithologies were generally low in magnetic minerals, all surveying methods are considered to be fit for purpose. A limited number of the Belvedere drillholes, namely BELKOPDD075, BELKOPDD076, BELKOPDD077, BELKOPDD082 and BELKOPDD090 only have down-hole surveys for dip but not azimuth, which is thought to have been caused by poor drillhole conditions.

Holes drilled by GMHL, Outokumpu, and GTK all have down-hole surveys at 10 m intervals for dip with a constant azimuth. The down-hole survey method is not recorded. Drillholes of less than 50 m were only surveyed at the start and end of the hole.

10.3 Collar Surveys and Casing

Collar surveys for drillholes drilled by Belvedere are conducted with a differential GPS (DGPS). Glenmore Highlands (GMHL) drill collars have been resurveyed by Belvedere for holes where the casing has been located. Drillholes where the casing was not found include KDD010, KDD012 and KDD015. The other GMHL holes have been surveyed with DGPS. The re-surveying of the GMHL holes has increased the confidence in the understanding of the spatial location of the drillholes, but this is not available for the Outokumpu and GTK drillholes.

10.4 Core Logging

The core was logged at the Company's logging facility. The core was logged by the relevant company staff at the time of drilling. The geological logging intervals were based on lithological variations and visual grade variations in the identified lithologies, and in addition, Rock Quality Designation (RQD) measurements were taken on the basis of the assay intervals (approximately every 1 m). All core logging data was recorded initially on paper and manually entered into Microsoft Excel spreadsheets and geological software for modelling.

10.5 Interpretation of Results

The Company's drilling results were used as a basis for the interpreted mineralization wireframes for copper and gold. These were interpreted and digitised by SRK using a combination of Leapfrog and Datamine software.

The main body in the Kopsa Cu-zone measures 660 m along strike in east-west direction, and has an east-northeast striking discontinuous limb that measures 550 m as well as a few smaller mineralised bodies. All together the maximum width, roughly perpendicular to the strike, for the whole package of bodies, is 670 m. The mineralization is interpreted as striking at 105º, dipping 20 - 30º towards the south.

The main features for the wireframe for the Au-zone are similar in extension, direction, and shape as the Cu-zone, though not always overlapping with the Cu-zone.

11 SAMPLE PREPARATION, ANALYSES, AND SECURITY

11.1 Introduction

A general description of sample preparation methods, assay techniques, and sample security procedures implemented by the Company at Kopsa is presented below. This covers all work completed to date on the Project.

11.2 Core Sampling

SRK notes that no documentation regarding the methodology or quality of the GTK and Outokumpu assaying is available. Consequently this data has not been used to derive the Mineral Resource estimate presented herein and is not discussed further in this report.

With regards the more recent drilling, during logging, the core was measured and sample intervals selected by staff geologists for sample analysis. The selected intervals were marked on the core and on the core boxes. Drill core was cut into two halves and sent to an accredited laboratory for sample preparation and analysis.

All mineralised core intersections from recent drilling have been sampled by the Company. The standard sample interval is 1 m, with some intervals shortened or lengthened to accommodate geological contacts. The sample intervals and numbers were either recorded onto paper and then entered into a Microsoft Excel spreadsheet, or entered directly into Microsoft Excel.

Drill core intersections observed during the site visit showed very good recovery and generally good quality core. Core loss is recorded systematically by the Company.

11.3 Chain of Custody& Security

The drill contractor has been responsible for transportation of the drill core from site to the Company's core archive and logging facility close to Hitura. Kopsa drill core and coarse rejects are stored in locked sheds at these facilities.

All project data are stored on the Company's mine site server at their adjacent Hitura mine, with appropriate data backup.

11.4 Sample Preparation

11.4.1 Introduction

The laboratories used over the life of the project are:

- Labtium (previously GTK Laboratory), in Kuopio, Finland;
- Labtium, in Rovaniemi, Finland; and
- ALS Chemex, in Öjebyn, Sweden.

11.4.2 Labtium

The laboratories at Kuopio and Rovaniemi are operated by the same company (Labtium) and use the same same preparation and assay methodology which is described below:

- The split drill core (max. weight 10 kg) is dried at 70°C;
- The samples are then crushed using a jaw crusher 70% passing 2 mm. The crusher is cleaned using compressed air between samples;
- The crushed sample is then split in a rotary splitter to provide a 0.8 to 1.5 kg sub-sample;
- The sub-sample is then pulverised using a LM2 pulverising mill;
- The rest of the crushed reject is bagged and labelled and returned to Belvedere for storage;
- The pulverising puck and the bowl of the LM2 are cleaned with glass beads between samples; and
- After pulverising the pulp is split into further sub-samples for assaying and archiving.

11.4.3 ALS Chemex

Sample preparation, for samples which weighed less than 3 kg at ALS Chemex comprised:

- The logging of the sample in the tracking system;
- weighing, drying, and coarse crushing of the entire sample; and
- pulverizing the entire sample to better than 85% passing 75 micron.

11.5 Sample Analysis

11.5.1 Labtium

The Company selected method 510P (conventional ICP - AES) for Cu and base metal analyses, along with the 705P (ICP - AES) method for Au. Details and detection limits are summarised in Table 11-1 below. Earlier exploration phases by Belvedere had also used multi-element ICP-AES packages by method 511P, similar to 510P, as well as method 704P which is an ICP –AES method for Au assays.

Table 11-1: Main methods and detection limits used at Labtium

Element / Oxide	Method	Units	Detection Limit
Ag	ICP AES (510P)	g/t	1
As	ICP AES (510P)	g/t	30
Cd	ICP AES (510P)	g/t	1
Co	ICP AES (510P)	g/t	1
Cr	ICP AES (510P)	g/t	5
Cu	ICP AES (510P)	g/t	3
Fe	ICP AES (510P)	g/t	100
Mn	ICP AES (510P)	g/t	1
Mo	ICP AES (510P)	g/t	5
Ni	ICP AES (510P)	g/t	3
Pb	ICP AES (510P)	g/t	10
S	ICP AES (510P)	g/t	100
Sb	ICP AES (510P)	g/t	20
Zn	ICP AES (510P)	g/t	1
Au	ICP AES (705P)	ppb	2

11.5.2 ALS Chemex

The prepared sample pulps were analysed at different ALS Chemex laboratories, usually that in Vancouver. The laboratory used the ME-ICP61 method (conventional ICP - AES) for Cu and base metal analyses. For gold analysis, the laboratory used the Au-AA24 (Fire assay) method in 2004, the Au-AA25 (Fire assay) method for Au post 2004, and Au-GRA22 for higher grade samples, where the assays were deemed to be over the detection limit with Au-AA24 and Au-AA25. The analyses carried out by ALS Chemex, including the methods and detection limits, are summarised in Table 11-2 below.

Table 11-2: Methods and Detection Limits for ALS assaying

Element / Oxide	Method	Units	Detection Limit
Ag	ICP (ME-ICP61)	g/t	0.5
Al	ICP (ME-ICP61)	%	0.01
As	ICP (ME-ICP61)	g/t	5
Ba	ICP (ME-ICP61)	g/t	10
Be	ICP (ME-ICP61)	g/t	0.5
Bi	ICP (ME-ICP61)	g/t	2
Ca	ICP (ME-ICP61)	%	0.01
Cd	ICP (ME-ICP61)	g/t	0.5
Co	ICP (ME-ICP61)	g/t	1
Cr	ICP (ME-ICP61)	g/t	1
Cu	ICP (ME-ICP61)	g/t	1
Fe	ICP (ME-ICP61)	%	0.01
Ga	ICP (ME-ICP61)	g/t	10
K	ICP (ME-ICP61)	%	0.01
Mg	ICP (ME-ICP61)	%	0.01
Mn	ICP (ME-ICP61)	g/t	5
Mo	ICP (ME-ICP61)	g/t	1
Na	ICP (ME-ICP61)	%	0.01
Ni	ICP (ME-ICP61)	g/t	1
P	ICP (ME-ICP61)	g/t	10
Pb	ICP (ME-ICP61)	g/t	2
S	ICP (ME-ICP61)	%	0.01
Sb	ICP (ME-ICP61)	g/t	5
Sr	ICP (ME-ICP61)	g/t	1
Ti	ICP (ME-ICP61)	%	0.01
U	ICP (ME-ICP61)	g/t	10
V	ICP (ME-ICP61)	g/t	1
W	ICP (ME-ICP61)	g/t	10
Zn	ICP (ME-ICP61)	g/t	2
Pb	ICP (Pb-OG62)	%	0.001
Ag	ICP (Ag-OG62)	g/t	1
Cu	ICP (Cu-OG62)	%	0.001
Zn	ICP (Zn-OG62)	%	0.001
Au	Fire Assay (Au-AA24)	g/t	0.005
Au	Au-GRA22	g/t	0.5
Au	Fire Assay (Au-AA25)	g/t	0.01

11.5.3 Laboratory Accreditation

Labtium has FINAS T025 accreditation ISO/IEC 17025:2005. According to FINAS, a laboratory's fulfilment of the requirements of ISO/IEC 17025:2005 means the laboratory meets both the technical competence requirements and management system requirements that are necessary for it to consistently deliver technically valid test results and calibrations. The management system requirements in ISO/IEC 17025:2005 are written in language relevant to laboratory operations and meet the principles of ISO 9001:2008 Quality Management Systems Requirements and are aligned with its pertinent requirements. This accreditation represents a higher standard than ISO 9001:2000. According to the website of Labtium, Labtium’s quality system fulfils the requirements of the Standards Council of Canada (CAN-P-1579), Guidelines for Accreditation of Mineral Analysis Testing Laboratories.

ALS is accredited by ISO 9001:2000 overall and conforms to the requirements of CAN-P-1579 and CAN-P-4E (ISO/IEC 17025:2005) by the Standards Council of Canada (SCC) for a number of specific test procedures.

SRK has visited the laboratory in Piteå and can confirm that the sample preparation is conducted to a high standard.

11.6 Quality Assurance and Quality Control (QAQC) Methodologies

11.6.1 Samples Submitted

The drilling and sampling procedures are supported by QAQC routines, which are consistent with industry best practice. Blanks were inserted by Belvedere at the beginning, and sometimes at the end, of each batch. Certified reference materials (CRMs or standards) were inserted by Belvedere every 20^{th} sample, and are purchased from CDN Resource Laboratories Ltd., with standard acceptable range ±10% of the defined value. A total of 8 different standards were used and their grades varied in between 0.71 - 9.98 g/t Au. Belvedere added no CRMs or blanks in their first drilling phases, but did include a total of 9 duplicates.

The laboratories used also added both blanks and Au CRMs as part of their own in-house QAQC protocols. Phase one and two correspond to 1,024 samples assayed in total. Laboratory blanks added corresponds to 5% of the assays and laboratory added CRMs corresponds to 6% of the samples.

For Belvedere´s drilling phases three to six, 7372 samples were analysed. Belvedere added 96 blanks and 472 standards corresponding to 0.6% and 3% respectively of the assays, while the laboratories added blanks corresponding to 3% of the samples and standards corresponding to 4%. No data is available for the QAQC procedures employed for the pre-Belvedere phases of exploration. SRK notes that only data from a single pre-Belvedere drilling phase, namely that conducted by Glenmore Highlands, was used in deriving the Mineral Resource estimate presented here.

11.6.2 Certified Reference Materials (CRMs) / Standards

Table 11-3 shows the CRM samples used at Kopsa all of which were purchased from CDN Resource Laboratories Ltd. The samples are intended as reference material for the determination of Au and in one case Au and Ag in the same reference. No Cu CRM was used in the sampling campaigns.

Table 11-3: CRMs used for Kopsa

CRM	Au (g/t)	Ag (g/t)
CDN-GS-3H	3.04 +/-0.23	
CDN_GS-10C	9.71+/-0.65	
CDN_GS-10D	9.50 +/-0.56	
CDN_GS-P7B	0.71+/-0.07	13.4 +/-1.6
CDN-GS-3F	3.10+/-0.24	
CDN-GS-12	9.98+/-0.37	
CDN-GS-P8	0.78+/-0.06	

Figure 11-1 shows the performance of the Belvedere CRMs for Au for the drilling campaigns between 2004 and 2007. Figure 11-2 shows how ALS Chemex in-house CRMs performed during the same period.

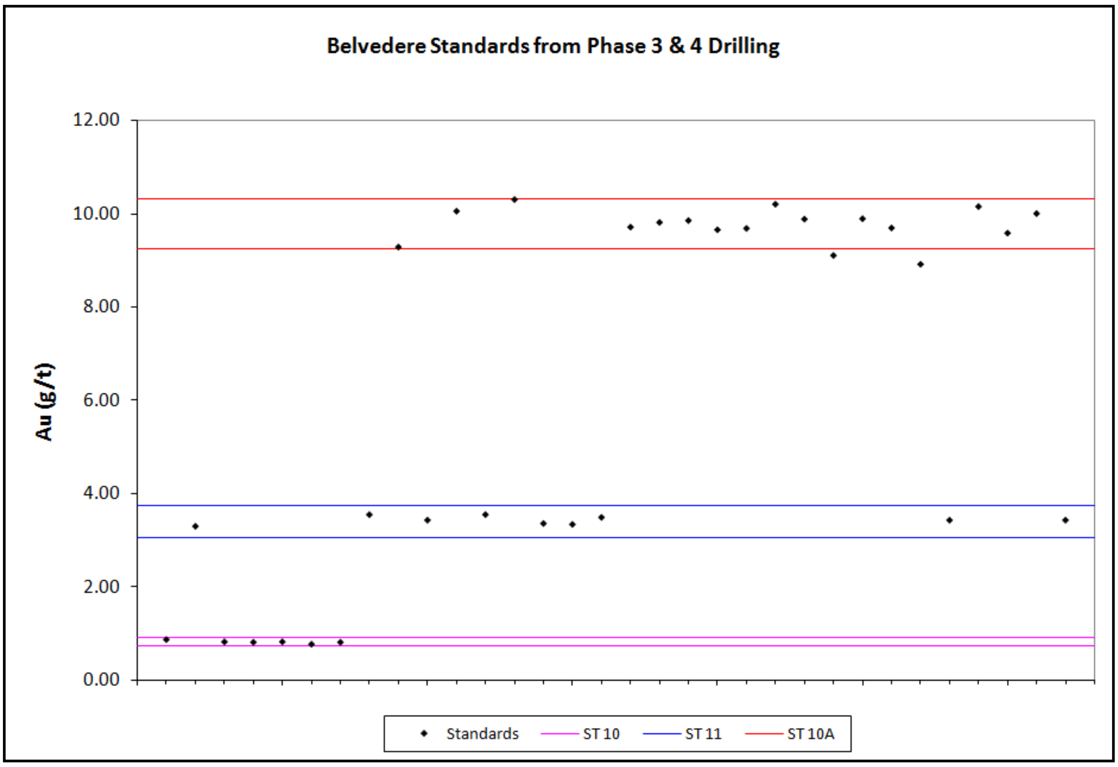


Figure 11-1: Belvedere Standards for Au for 2004-2007(Source: SRK, July 2013).

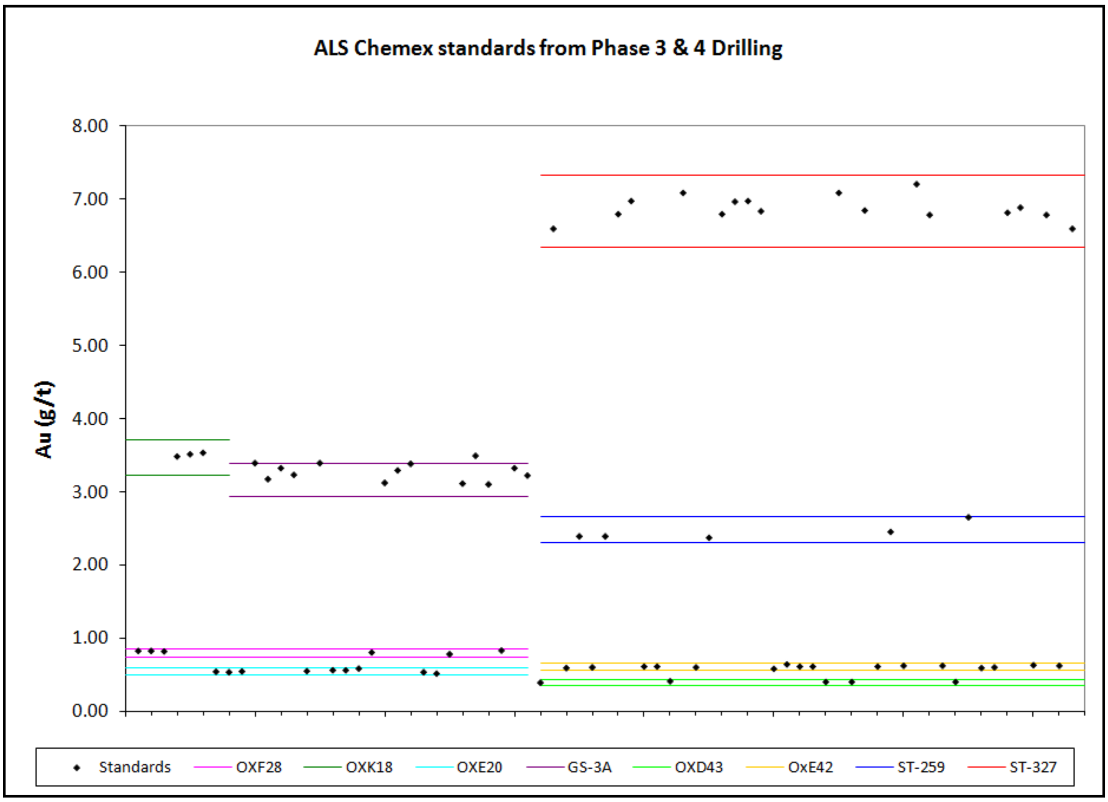


Figure 11-2: ALS Chemex standards for Au for 2004-2007 (Source: SRK, July 2013).

Figure 11-3 shows the performance of the Belvedere CRM for Au for the 2010 drilling campaign. Figure 11-4 shows how the Labtium in-house standards performed during the same period.

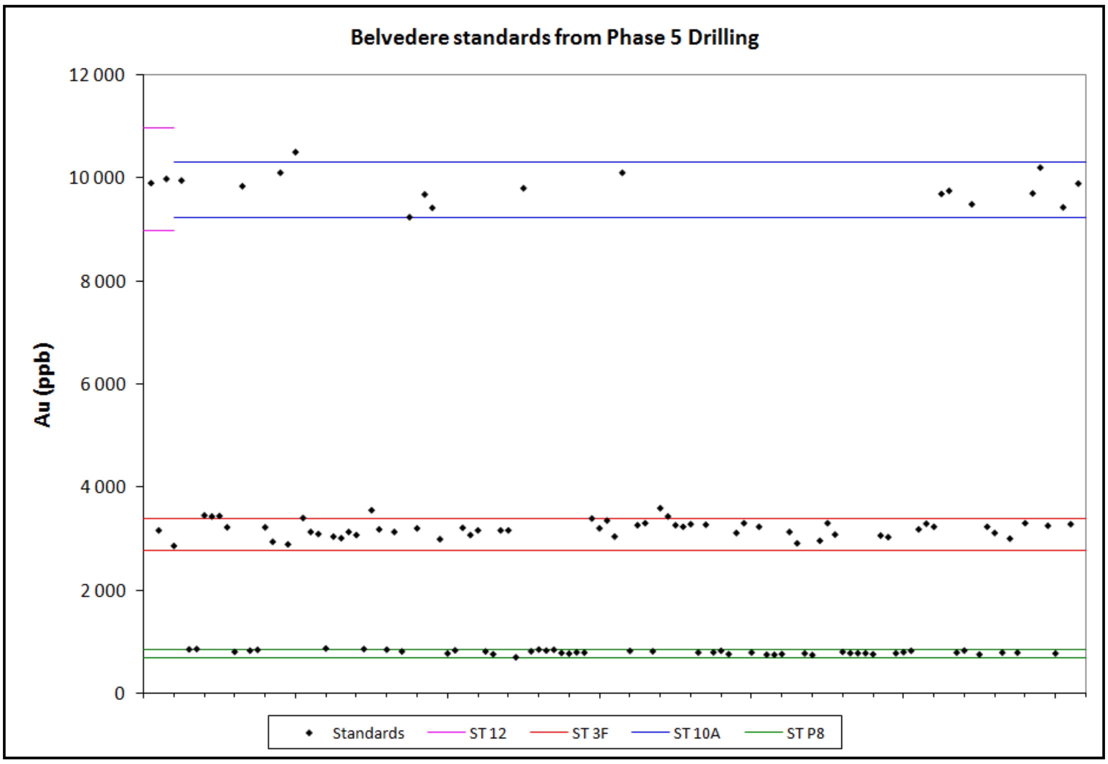


Figure 11-3: Belvedere standards for Au for 2010 (Source: SRK, July 2013).

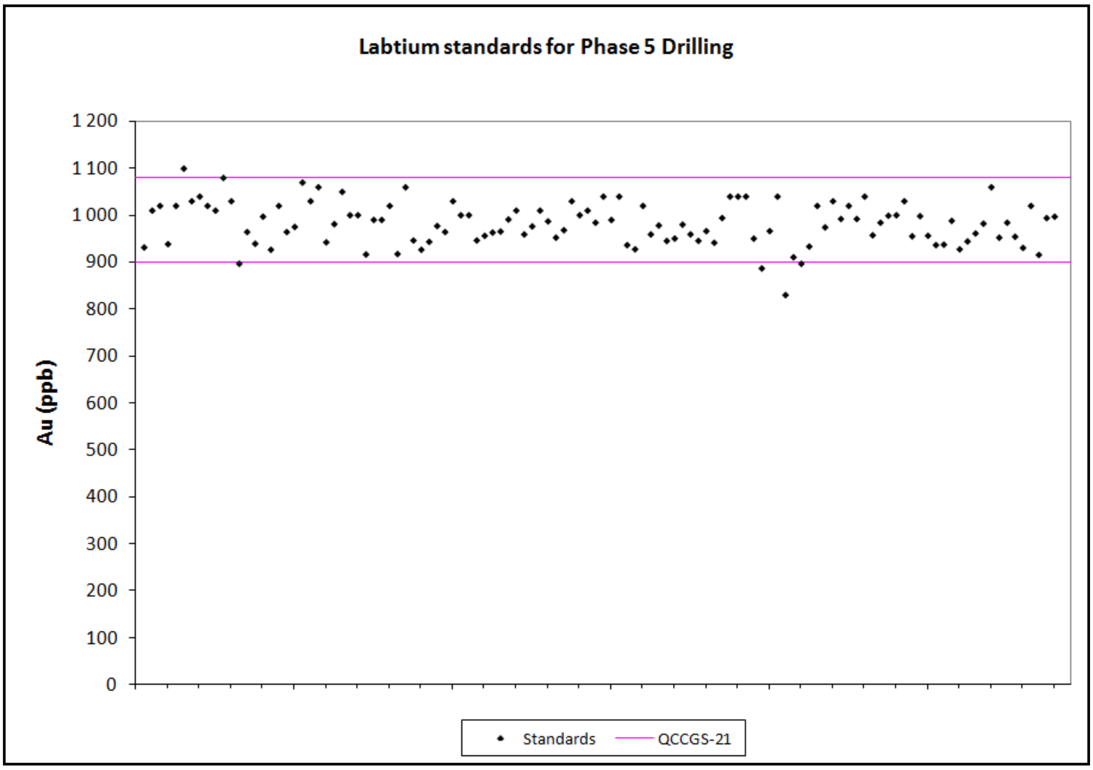


Figure 11-4: Labtium standards for Au for 2010 (Source: SRK, July 2013).

Figure 11-5 shows the performance of the Belvedere CRM for Au for the 2011 drilling campaign. Figure 11-6 shows how the ALS Chemex in house CRMs performed during the same period.

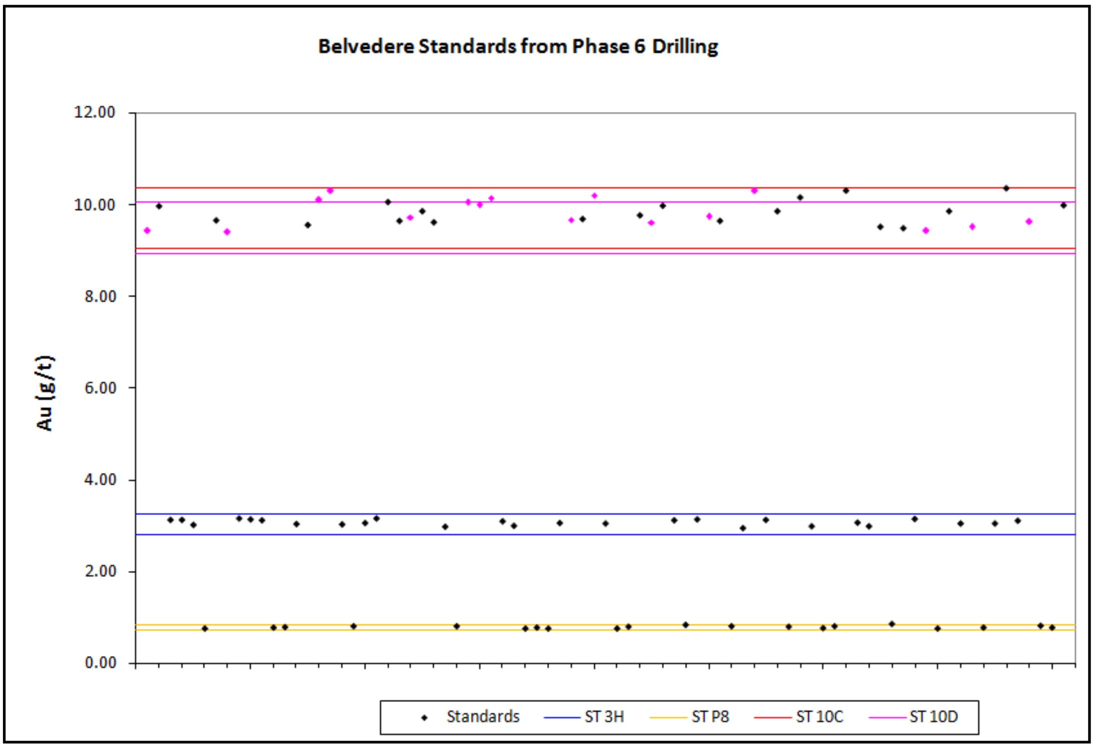


Figure 11-5: Belvedere standards for Au for 2011 (Source: SRK, July 2013).

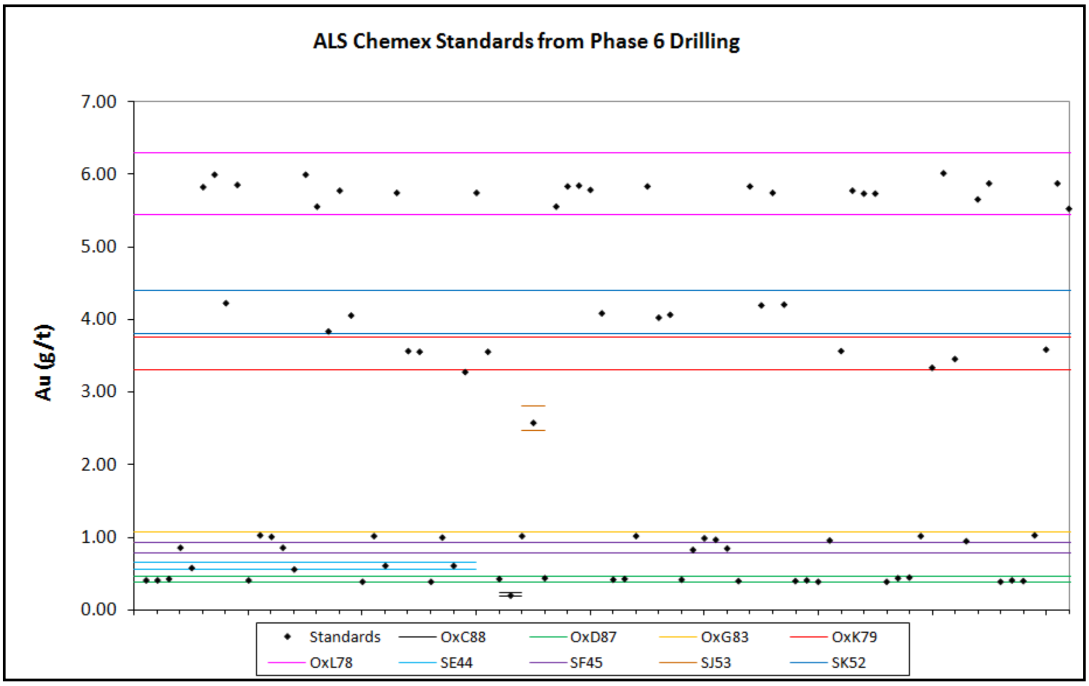


Figure 11-6: ALS Chemex standards for Au for 2011 (Source: SRK, July 2013).

Between 2003 and 2011, the majority of results for Belvedere CRMs, from both laboratories, lie within of the certified grade ranges. There does not appear to be a bias over time and the results appear to be evenly distributed about the recommended mean grade.

11.6.3 Duplicates

Figure 11-7 to Figure 11-9 show the results of the laboratory duplicates for Au (g/t) for ALS and Au (ppb) for Labtium for the drilling campaigns between 2004 and 2011. The duplicate samples show a strong correlation to the original sample and SRK considers that the sample preparation and analysis shows an acceptable level of repeatability.

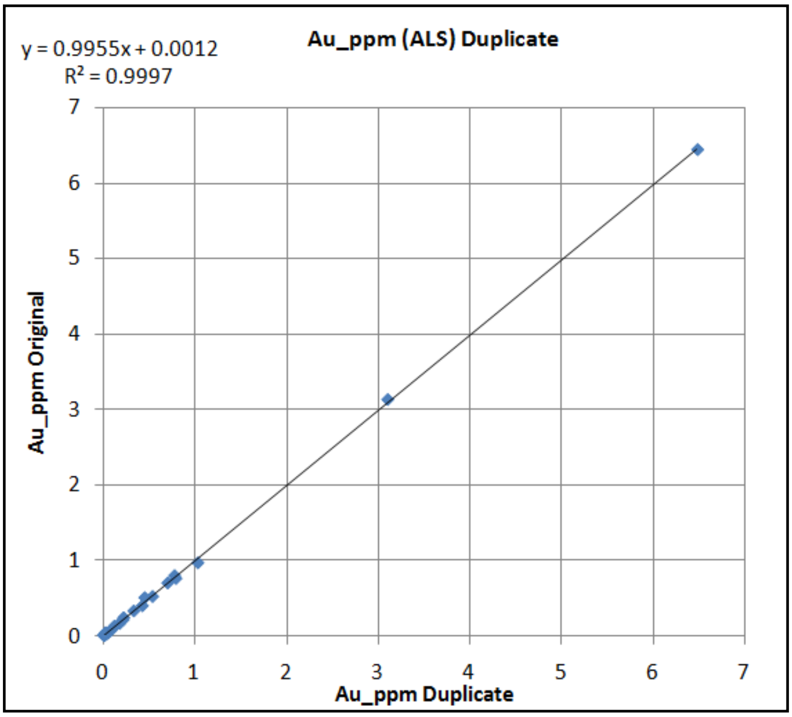


Figure 11-7: Duplicates Au (g/t) ALS 2004-2007 (Source: SRK, July 2013).

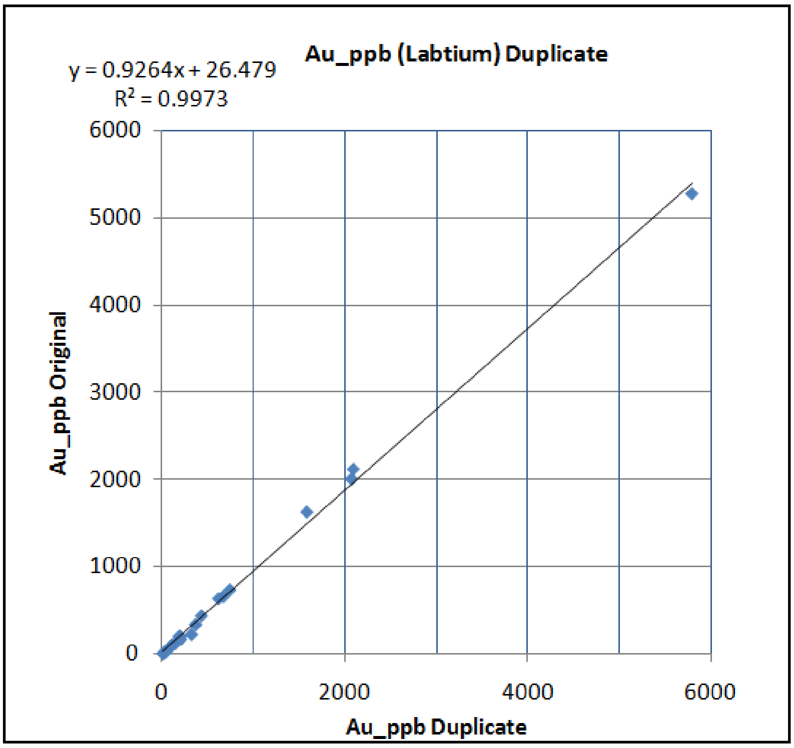


Figure 11-8: Duplicates Au (ppb) Labtium 2010 (Source: SRK, July 2013).

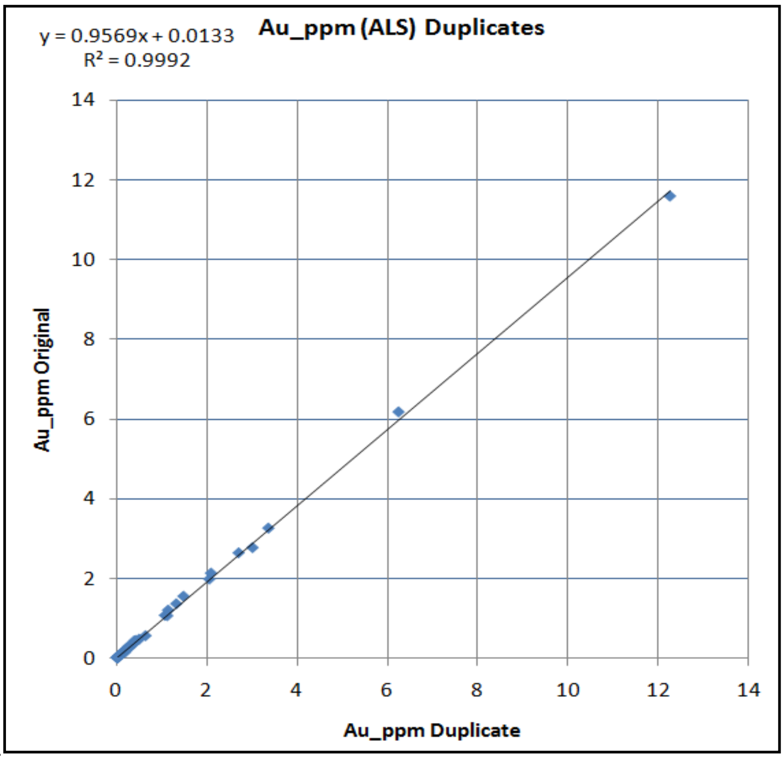
1 

Figure 11-9: Duplicates Au (g/t) ALS 2011 (Source: SRK, July 2013).

11.7 Density Measurements

The bulk density of the mineralization was modelled from the specific gravity data collected by the Company's geologists using simple digital scales and the Archimedes method. Specific gravity was measured using 100 to 200 mm sections of intact drill core. Over 1 650 measurements within the mineralised zone were taken, resulting in an average bulk density of 2.73 g/cm^3.

11.8 SRK Comments

SRK considers that the Company has developed logging and sample preparation procedures that enable the appropriate handling of drill core from the rig through to sample selection, logging and data collection and dispatch of cut samples to the preparation laboratory. SRK considers the core logging facilities to be housed in a suitable building which is clean, modern and appears to be well-managed. Appropriate security procedures are in place and the assaying has been carried out using appropriate techniques and by qualified laboratories. SRK would however recommend that the Company submit a proportion of future samples to an umpire laboratory and also that duplicate samples form part of the Company's standard quality assurance protocols. Further, SRK recommends that appropriate Cu standards be introduced in addition to current gold standards.

SRK is of the opinion that the assay and density data provided by Belvedere is of sufficient quality to support the Mineral Resource estimate presented in this report.

12 DATA VERIFIATION

In order to independently verify the Company's drill database, during the site visit, SRK carried out:

- An inspection of several drill collars at the Kopsa site to confirm location of these;
- Drill core inspection of nine Belvedere holes with good spatial representation across the deposit, cross-checking geology, mineralization, sample interval and sample numbers against the Company's drill database; and
- Collection of 44 coarse reject samples for check assaying. These samples were selected by SRK on the basis of their spatial and temporal representivity.

12.1.1 SRK Check Assaying

SRK selected 20 coarse reject samples from 8 drillholes for assay at Labtium, and 24 samples from 7 different drillholes for assay at ALS Chemex. These samples were assigned new sample numbers before being sent to the laboratories. Samples originally assayed at Labtium were sent to ALS Chemex for check assaying and vice versa.

The samples sent to Labtium were re-assayed by method code 705P for Au and 511P for multi-element analysis ICP_OES. The samples that were sent to ALS were re-assayed by method Au-AA25 for Au and ME-ICP61 for multi-element analysis.

12.1.2 Check Sample Sent to Labtium

Table 12-1 details assay results received by SRK for 20 samples sent to Labtium for analysis by method codes 705P and 511P. The corresponding original assay results are presented for comparison.

Figure 12-1 and Figure 12-2 compare the results of SRK's duplicate sample analyses for Cu (ppm) and Au (ppb) carried out at Labtium by method 705P and 511P with the Company's original results. There is clearly a strong correlation between the originals and the duplicates and also the overall mean is very close.

The only outlier to this was a sample where the duplicate assay was 1,700 ppm Au (ppb), as compared to an original assay of 920 ppm Au (ppb).

Table 12-1: Details of SRK Duplicate samples sent to Labtium for Cu (g/t) and Au (ppb) with respect to original assay results

Sample ID	Cu_g/t (SRK)	Au_ppb (SRK)	Cu_g/t (Original)	Au_ppb (Original)
4872	2270	9360	2220	9560
4873	697	538	665	550
4892	6090	2420	5980	2370
5112	770	284	783	280
5193	6170	505	6420	520
5194	4110	1750	4110	1830
5461	3210	4800	3180	4730
5495	1950	1810	1885	1460
5623	2510	2140	2630	2080
5642	520	748	522	710
5718	6550	1020	6350	890
5753	715	2190	679	2640
5788	1670	2930	1650	2770
5798	635	2380	655	2130
5888	597	280	629	250
5926	971	4020	900	3810
5949	2210	406	2150	420
5999	472	1700	465	920
6024	3770	2860	3700	2910
6054	621	433	617	420

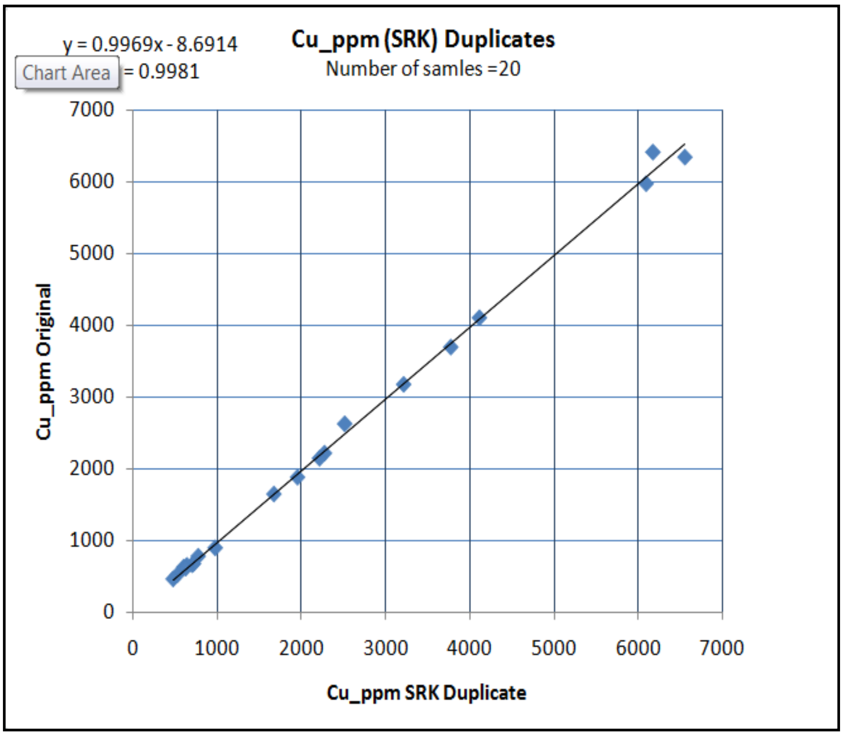


Figure 12-1: Cu_g/t SRK Duplicate Samples against Originals sent to Labtium (Source: SRK, July 2013).

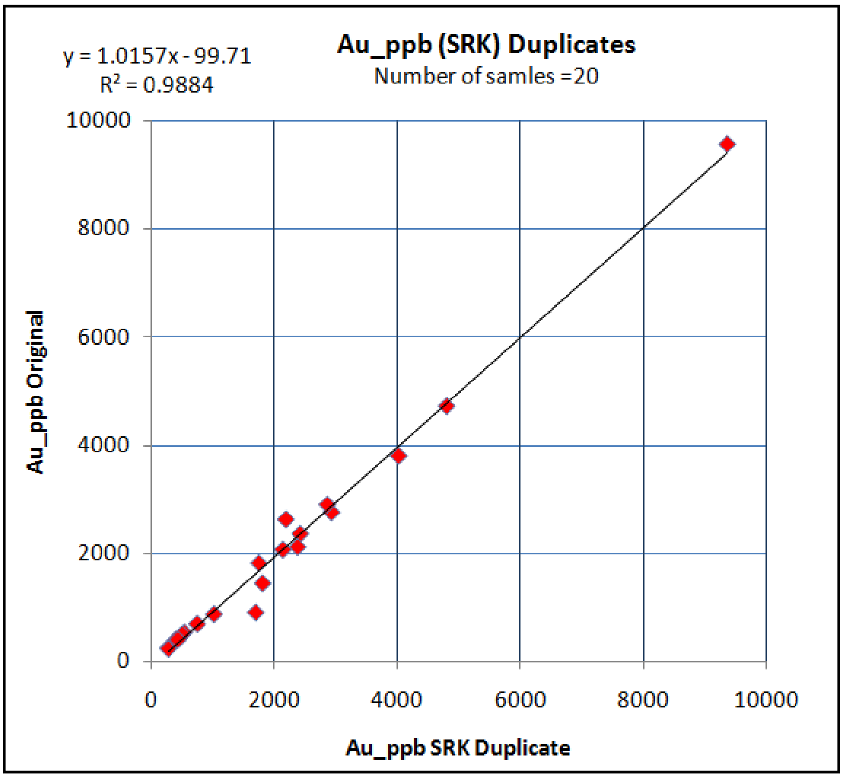


Figure 12-2: Au_ppb SRK Duplicate Samples against Originals sent to Labtium (Source: SRK, July 2013).

12.1.3 Check Sample Sent to ALS Chemex

Table 12-2 details assay results received by SRK for the 24 samples sent to ALS Chemex for analysis by method codes Au-AA25 and ME-ICP61.

Figure 12-3 and Figure 12-4 compare the results of SRK's duplicate sample analyses for Cu (ppm) and Au (ppm with the Company's original analyses. Again there is a very good correlation between the two datasets.

Table 12-2: Details of SRK Duplicate samples sent to ALS for Cu_g/t and Au_ppm with respect to original assay results

Sample ID	Cu_g/t (SRK)	Au_ppm (SRK)	Cu_g/t (Original)	Au_ppm (Original)
BKD0049	386	0.86	355	0.795
BKD0124	856	1.02	827	0.995
BKD0169	2070	0.18	2050	0.144
BKD0175	6220	0.70	5500	0.712
11K1982	1125	1.18	1270	1.26
11K1986	945	0.77	937	0.753
11K1992	1530	0.80	1610	1.04
11K1998	3910	1.75	4090	1.39
11K2001	2500	0.38	2460	0.379
11K2027	2460	0.40	2550	0.348
11K2028	3440	0.56	3600	0.526
11K2049	1070	0.43	1100	0.454
11K2052	1190	8.46	1220	7.89
11K2128	1095	0.12	1150	0.129
11K2133	1090	0.81	1100	0.81
11K2139	717	0.40	749	0.449
11K2170	1110	0.42	1190	0.455
11K2197	3510	1.85	3610	2.41
11K2198	5850	0.67	6070	0.699
11K2205	1885	1.09	1980	1.13
11K2277	1400	0.84	1450	0.932
11K2291	3260	7.74	3420	8.01
11K2292	3560	2.25	3750	2.44
11K2298	741	0.97	735	1.03

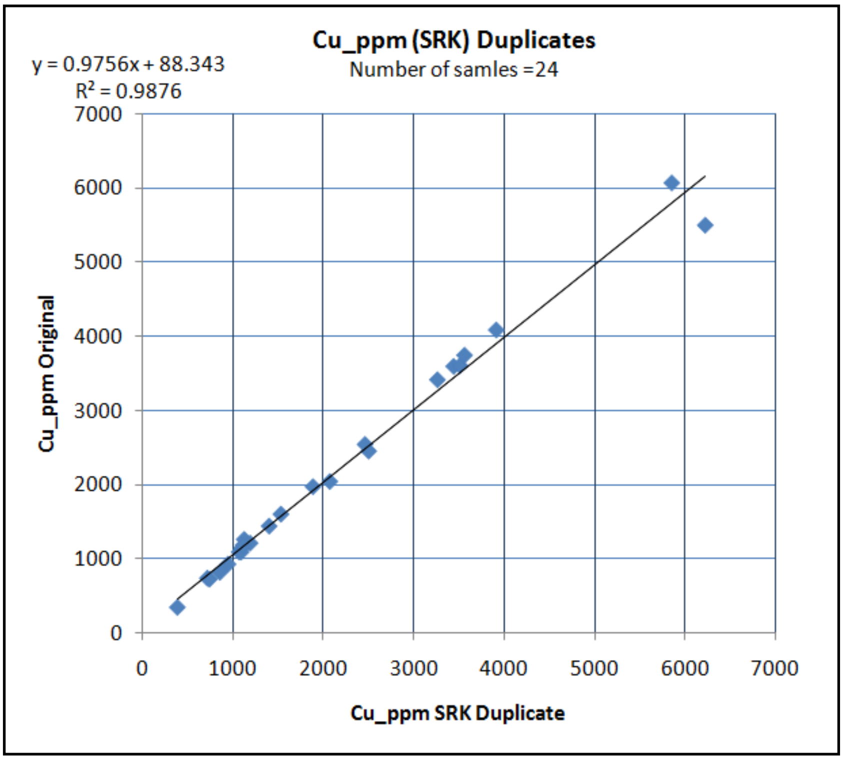


Figure 12-3: Cu_g/t SRK Duplicate Samples against Originals sent to ALS (Source: SRK, July 2013).

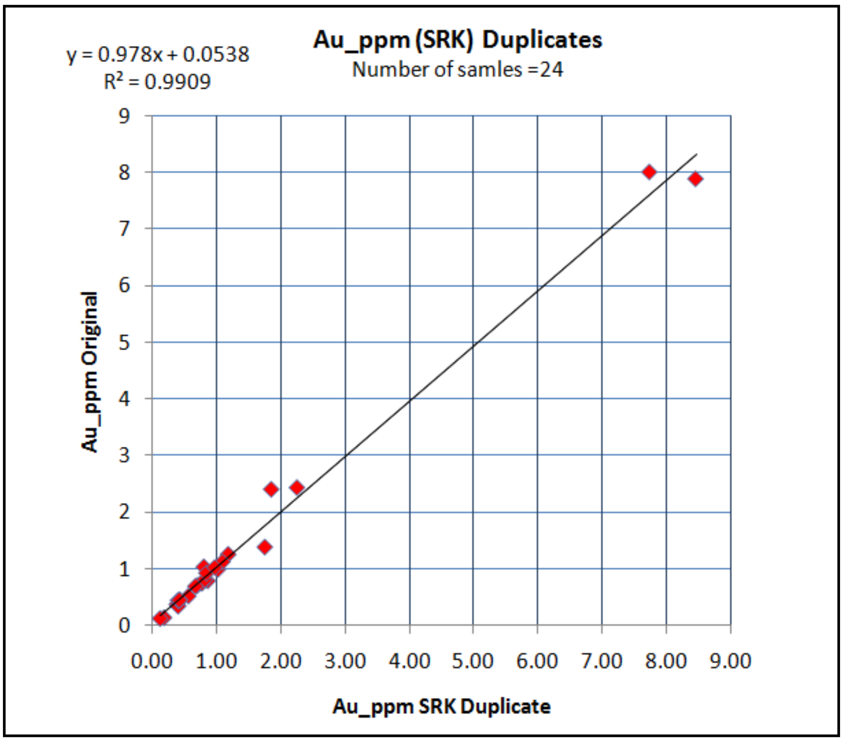


Figure 12-4: Au_ppm SRK Duplicate Samples against Originals sent to ALS (Source: SRK, July 2013).

12.2 SRK Comments

The number of collars located in the field, drill cores reviewed and check samples selected for assay by SRK represents a small proportion of the overall number of drill collars and analysis carried out on the Project as a whole. Notwithstanding this, no material errors were found during the course of these checks and this adds confidence in the Company's drillhole database and the repeatability of the assay methods used.

13 MINERAL PROCESSING AND METALLURGICAL TESTING

13.1 Historical Testwork

13.1.1 Introduction

Several metallurgical testwork programmes have been undertaken in support of the Kopsa Project since 2005.

13.1.2 McLelland Laboratories, USA, 2005

Testwork was undertaken on diamond drill core samples from two Glenmore Highlands drillholes at the McLelland Laboratory, Reno, Nevada, in 2005. The drillholes were KDD-1 and KDD-12. One composite sample was made for each hole, as well as a Master Composite from both holes combined.

The principal aim of the testwork programme was to test the amenability of the samples to gold recovery by cyanidation.

The head assays of the samples are shown in Table 13-1.

Table 13-1: McLelland Lab Samples Head Assays

Element	Unit	KDD-1	KDD-12	Master Composite
Au	g/t	2.67	1.22	1.58
Cu	%	-	-	0.21
Ag	g/t	-	-	2
Fe	%	-	-	3.07
S	%	-	-	0.85
Sulphide S	%	-	-	0.72
As	%	-	-	0.61
Organic C	%	-	-	<0.01

A mineralogical investigation conducted on the Master Composite observed the major sulphide minerals to be pyrite and chalcopyrite, with minor marcasite and arsenopyrite. No visible gold was observed.

A Bond Ball Mill Work Index (BWi) test was conducted on the Master Composite, with a BWi value at a closing screen size of 100 μm of 15.2 kWh/t.

Cyanide leach tests were conducted using mechanically stirred vessels, at a range of grind sizes: 80% -212 μm, 80% -75 μm and 80% -45 μm (all samples) and 80% -150 μm and 80% -106 μm (Master Composite only). The leach tests were conducted for a total period of 72 hours.

The results of the cyanidation tests are illustrated in Figure 13-1, which shows the recovery achieved for each grind size for each of the samples.

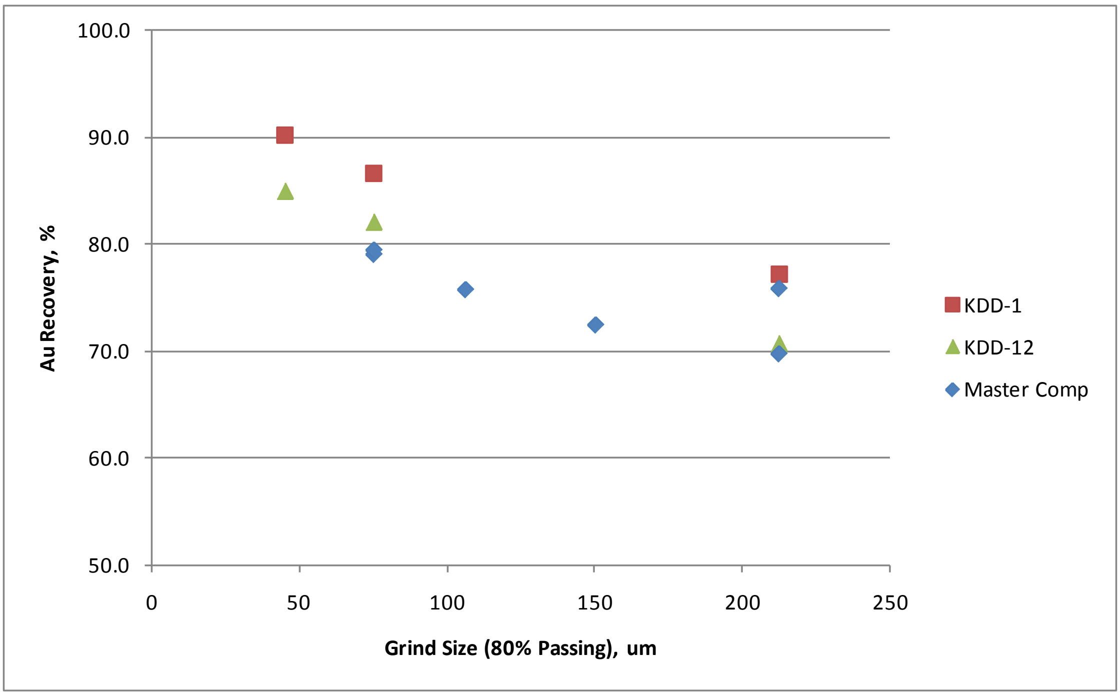


Figure 13-1: McLelland Lab Au Cyanidation Recoveries (Source: SRK, July 2013).

The cyanidation tests results show an increase in Au recovery with decreasing grind size, although it is not clear why the performance of the Master Composite was slightly inferior to that of the individual samples. The leach kinetics indicated that leaching was essentially complete after 12 hours (KDD-12 and Master Composite) and 24 hours (KDD-1). Cyanide consumptions ranged from 0.60 to 2.48 kg/t, with a general increase with decreasing grind size. Lime consumptions ranged from 0.9 to 4.9 kg/t. The level of repeatability in Au head grade between the initial head assays, test sample head assays and test recalculated values indicated negligible presence of coarse free gold.

13.1.3 GTK, Finland, 2008

Some testwork is briefly reported as having been undertaken by GTK and GSF Outokumpu in 2008. The GTK work reports on the construction of a composite sample from Belvedere drillholes BDD001, BDD002, BDD008 and BDD009. The composite assayed 1.77 g/t Au and 3.1 g/t Ag. This sample was subsequently sent to SGS to be used in the 2011 and 2012/2013 tests.

The GSF Outokumpu work reports on flotation testwork conducted on a sample blasted from the outcrop at Kopsa. This sample had an elevated Au content (4.92 g/t) but the testwork reported a recovery of 71% of the Au into a concentrate assaying 100 g/t Au. The concentrate also reported a Cu recovery of 88%, however the Cu content of the concentrate was low (3.44%). The As content of the concentrate was very high (approximately 15%). The flotation tailings reported a low As content (0.09%).

13.1.4 SGS Mineral Services, UK, 2011

Testwork was conducted in 2011 at the SGS Mineral Services laboratory in Cornwall, UK. The test was conducted on a single composite sample made up from material from drillholes BELDD001, BELDD002, BELDD008 and BELDD009. The sample assayed 1.80 g/t Au, 3.5 g/t Ag, 0.16% Cu, 0.86% As, 9.39% Fe and 0.87% S.

The Au head assay was determined using a Screen Fire Assay, and the lack of upgrading to the +106 μm fraction during this assay led SGS to conclude that there is negligible gravity recoverable gold.

Flotation testwork was initially aimed at the production of a bulk sulphide concentrate. Initial testwork focussed on the flotation response with grind size, the conclusions of which were that a fine grind size (80% -45 μm) was warranted, with the recovery of Au more sensitive to grind size than the recovery of Cu.

Reagent optimisation was conducted with the aim of maximising the Cu and Au recovery. At the latter end of this testwork, As assays were conducted, and the results showed that in addition to high Cu and Au recoveries, the As recovery was also very high. Differential flotation was then tested, and the results – at rougher stage – indicated the potential to float Cu separately from the As and the majority of the Au.

Differential flotation was next attempted at a cleaner stage, using the bulk sulphide rougher concentrate. While some selectivity was achieved, no significant upgrading of the Cu (i.e. to a marketable grade) was achieved from the bulk sulphide rougher concentrate – the highest Cu grade reported in the entire testwork programme was 9.3%; the highest As grade reported was 29.7%.

Cyanidation of the bulk sulphide rougher concentrate achieved an Au recovery of just below 50%; with low Cu and negligible As recoveries.

13.1.5 Comex, Norway, 2011

Sorting is being considered as a means of reducing the amount of material that has to be transported between the Kopsa mine site and the Hitura process plant.

A sample of Kopsa material was sent to the Comex facility in Rud, Norway, where optical sorting was tested on material within the size range 50 – 150 mm. The sorting criteria was based on the presence of quartz as the valuable mineral indicator.

The sorting trial produced a 100% quartz product, i.e. all of the product particles contained quartz vein material, however the recovery of quartz-containing particles to that product was only 79%. One factor contributing to the rejection of quartz-containing particles was that the quartz veins in those particles were not visible to the single optical detector due to the way the particles were presented.

No mineral assays, e.g. Au, Cu A, S etc, were conducted.

13.1.6 GTK, Finland, 2012

A mini pilot flotation plant trial was conducted in early 2012 at the GTK Mintec facility in Outokumpu, Finland, using 190 kg of Kopsa mineralised material. The material was sourced from the following diamond drillholes: 34, 39, 42, 45, 49, 53, 56, 57, 64, 56, 72, 74, 75, 80, 82, 83, 85 and 86. The composite used in the testwork assayed 1.20 g/t Au, 0.18% Cu, 3.6 g/t Ag, 0.69% As and 0.79% S.

Laboratory scale flotation tests were used to establish the process conditions to maximise recovery, at the target grind size of 80% -45 μm (the actual grind size achieved was 82% -45 μm). The optimised laboratory test conditions produced recoveries of Au, Cu, Ag, As and S ranging from 95% to 99%, at a mass recovery of 19%.

The mini pilot plant was run at a feedrate of 15 kg/hr, and over the course of the run produced approximately 20 kg of rougher concentrate assaying 4.39 g/t Au, 13.3 g/t Ag, 0.87% Cu, 2.5% As and 3.7% S. The recoveries achieved ranged from 76% for Ag to 98% for As, at a mass recovery of 21%. One of the reasons suggested for the inferior mini pilot performance compared to the laboratory results was that the pilot plant grind size was slightly coarser (75% - 45 μm) than the grind size achieved in the laboratory tests.

The flotation tailing from the pilot plant trial assayed <0.01% As.

13.2 Current Testwork Programme

13.2.1 Introduction

A further programme of metallurgical testwork has been undertaken in support of the PEA. This work has been undertaken either by, or under the auspices of, SGS Mineral Services, UK.

13.2.2 2011 Diamond Core

Further flotation testwork has been undertaken using the diamond drill core left over from the 2011 SGS testwork programme.

The aims of this flotation programme have been to:

- Produce a high grade Cu concentrate with low As;
- Produce a high Au recovery concentrate; and
- Produce a low As tailings.

The testwork has been conducted at batch and locked cycle stage.

Two mineralogical studies were conducted; one focussing on the sulphide minerals, and the other on gold. The sulphide mineral study, which was undertaken on a sample that had been ground to approximately 80% -53 μm showed high liberation of the sulphide minerals at that grind size. Arsenopyrite was the major sulphide mineral (2.4%) followed by iron sulphides (0.9%) and copper sulphides (0.8%). The average grain size (i.e. 50% passing size) of the sulphide minerals ranged from 26 μm for the copper sulphides to 34 μm for the iron sulphides. The gold deportment study indicated that 18% of the gold was liberated (average particle size 24 μm), 18% was exposed (average particle size 8 μm) and 61% was locked (average particle size 2 μm). The largest particle dimension observed was 154 μm for a liberated particle.

The best locked cycle test result reported to date is summarised in Table 13-2.

Table 13-2: Locked Cycle Test Results

Stream	Wt (%)	Assay					Distribution (%)				
		Cu (%)	As (%)	S (%)	Au (g/t)	Ag (g/t)	Cu	As	S	Au	Ag
Cu Con	0.70	19.9	4.17	27.4	123	283	87.9	4.5	19.9	47.7	65.8
Au Con	2.94	0.26	18.5	14.8	22.1	20.2	4.8	83.5	45.4	36.2	19.8
Tail	96.8	0.01	0.03	0.28	0.26	0.46	8.2	3.8	28.2	14.2	14.9

Further batch testwork has been undertaken aiming to increase the Cu grade and decrease the As grade of the Cu concentrate. The best results reported to date are summarised in Table 13-3.

Table 13-3: Cu Cleaner Test Results

Test No	Wt (%)	Assay					Distribution (%)				
		Cu (%)	As (%)	S (%)	Au (g/t)	Ag (g/t)	Cu	As	S	Au	Ag
FT6	0.19	24.8	2.31	32.4	86.0	242	28.1	0.6	6.9	8.0	15.7
FT7	0.46	21.6	3.70	30.9	104	207	60.4	2.4	17.3	23.9	30.7
FT8	0.27	25.6	0.87	32.4	56.1	204	47.7	0.3	9.7	8.0	18.9
FT9	0.53	21.9	1.87	32.0	112	263	82.8	1.3	18.4	10.7	18.8
FT11	0.15	26.5	0.08	34.9	72.8	368	47.3	0.02	6.0	37.7	19.0
FT12	0.22	26.4	0.12	34.1	127	267	36.2	0.04	6.1	14.8	17.8
FT14	0.35	26.0	0.39	16.4	142	285	57.9	0.2	4.7	19.6	21.5
FT15	0.42	24.8	0.30	33.0	101	276	66.9	0.2	10.6	15.0	39.9

Cyanidation tests have been undertaken on the Au concentrate generated from a later locked cycle test (described in Table 13-2). The tests were conducted under intensive cyanidation conditions, principally with respect to the free cyanide concentration, which was set at 10 g/l (i.e. 10,000 g/t). The initial test was conducted on the sample from the flotation test without further regrinding; this sample reported Au recoveries averaging 84% after 4 hours of leaching, with Ag recoveries averaging 64% after 6 hours and with a Cu dissolution of just over 40%. Regrinding the sample to 80% -20 μm improved the Au recovery (up to an average of 92% after 4 hours, with a slight increase in Ag recovery (from 60% after 2 hours to 70% after 48 hours) and an increase in Cu dissolution (from 40% after 4 hours increasing to approximately 55% after 48 hours). A further test was conducted under conditions intended to mimic a Gekko Systems In-Line Leach Reactor; this test was conducted on material that was not reground, and produced a slightly higher Au recovery (94.6% after 4 hours) and similar Ag and Cu extractions, however in this test all recoveries decreased after 4 - 6 hours of leaching (up to the 24 hour duration of the test).

A BWi was been determined for this material; at a closing screen size of 53 μm the BWi was 20.0 kWh/t.

13.2.3 2013 Outcrop Sample

A bulk sample was blasted from the outcrop at Kopsa for sorting testwork. The sample, which was taken in February 2013 and consisted of approximately 50 t, was crushed to a nominal -150 mm on site and trucked to the Hitura process plant site for splitting. A sub-sample of approximately 6 t was split out for testwork. Sorting testwork was conducted at Tomra in Hamburg, Germany. Some of this material was also sent to SGS.

Based on grab samples taken on site and on the sorting results, this sample was found to be both high in Au (approximately 2.0 g/t) and low in Cu (approximately 0.06%).

The sorting testwork first considered optical sorting (i.e. by colour difference), however the only distinguishing colour that could be identified was the orange / red oxidation of fracture surfaces. While this resulted in a separation for the +40 mm fraction, the subsequent assays showed that there had been no differentiation between the two fractions in terms of the minerals of interest.

Testwork on the -40 mm material was thus conducted using X-Ray Sorting. The -40 mm material was divided into narrower size fractions (-40+32, -32+20, -20+12 and -12+8) for the testwork. For each size fraction, an initial pass was followed by re-processing the reject material from the first pass.

The assays of the sorting fractions are summarised in Table 13-4.

Table 13-4: Sorting Test Results

Size Fraction (mm)	Sorting Method	Sample	Fraction Wt (%)	Assay			
				Cu (%)	S (%)	Au (g/t)	Ag (g/t)
+40	Optical	Product	43.4	0.054	0.44	1.10	1.0
		Reject	56.6	0.052	0.46	1.12	1.4
-40+32	X-Ray	Product	37.2	0.070	0.95	4.53	2.9
		Reject	62.8	0.055	0.34	0.42	0.6
-32+20	X-Ray	Product	37.3	0.078	1.13	4.74	2.3
		Reject	62.7	0.052	0.32	0.59	0.4
-20+12	X-Ray	Product	33.5	0.095	1.49	7.92	4.8
		Reject	66.5	0.054	0.31	0.39	1.6
-12+8	X-Ray	Product	27.9	0.093	1.61	9.49	1.8
		Reject	72.1	0.059	0.34	0.97	0.6
-8			-	0.100	1.27	3.97	3.1
Total				0.067	0.55	1.79	1.5

The X-Ray sorting testwork exhibited a significant upgrading of Au, and to a lesser extent Ag, into the Product, with only a minor upgrading of Cu. This may suggest a mineral different association between the sulphide species, e.g. arsenopyrite (containing Au) and Cu, and/or may be a reflection of the slightly finer average particle size of the Cu minerals.

Excluding only the X-Ray Reject material, i.e. including both all of the +40 mm fraction and the -8 mm fraction, the ore sorting testwork rejected just under 40% of the overall mass while recovering 91% of the Au, 83% of the Ag and 67% of the Cu. The calculated assay of this material is 0.07% Cu, 2.99 g/t Au and 2.11g/t Ag.

Considering just the size fractions that were subjected to XRT sorting, i.e. -40 + 8 mm, XRT sorting resulted in a mass rejection of 64% while retaining 86% of the Au. The corresponding losses of Cu and Ag were 56% and 29% respectively. The calculated assay of this material is 0.08% Cu, 5.26 g/t Au and 2.94g/t Ag.

The products from the sorting testwork have been returned to SGS for downstream testwork. This has initially focussed on flotation on the combined XRT Product fraction. This material has also been submitted for a BWi determination.

Initial flotation testwork reported by SGS has indicated that the XRT Product has performed similarly to the material previously tested, with respect to rougher (at this stage) grades and recovery, given the difference in Cu and Au head grades between the XRT Product and the whole sample.

The BWi for the XRT Product sample was 22.7 kWh/t.

13.3 Recommendations

While the testwork conducted to date has indicated the potential to produce a marketable copper concentrate and a final tailing low in arsenic (and other sulphides), the production particularly of the copper concentrate has been difficult to execute at laboratory scale due to the low Cu head grade of the material.

Therefore, as part of the next phase of the project's development, SRK recommends that some flotation testwork is undertaken at a pilot plant scale, in order to account for the low volume (with respect to the head) of copper concentrate produced. In addition, testwork at this scale will be required in order to price sufficient quantity of concentrate to provide samples for market testing (i.e. customer smelting tests).

Further developmental testwork is also required for the sorting option, and again such testwork is best undertaken at pilot scale. Pilot flotation testwork should be undertaken both on the product from sorting, and also on "unsorted" material.

In addition to pilot scale testwork, laboratory scale testwork should also be undertaken on a range of samples that cover the expected variability within the deposit, in terms of head grade, mineralogy, depth and lateral extent.

Particularly with regard to the sorting stage, given the sorting method chosen on the basis of the recent testwork, i.e. XRT sorting, and given the corresponding maximum particle size for his method (32-40 mm), it should be possible to make use of diamond drill core for this testwork, i.e. there seems no need to take a bulk sample via trenching or a "test pit" in order to provide "fresh" broken rock (as would probably be required for colour sorting, which is much more reliant on the surface properties of the rocks).

The cost for such a metallurgical testwork programme to support the next phase of the project's development is likely to be of the order of EUR 0.5-1.0 million.

14 MINERAL RESOURCE ESTIMATES

14.1 Introduction

SRK has produced a Mineral Resource estimate of the Kopsa deposit using the data from both the historic drilling (specifically, certain holes from the Glenmore Highlands drill campaign) and the Company's drilling programmes. A database was compiled using data from 126 diamond drillholes, with collar, survey, geological and assay information, containing a total of 14,324 m of drilled metres. In the process of completing the resource estimate, SRK validated and verified the database, interpretation and available data. The block dimensions selected for the block model was 10 m x 10 m x 5 m, which reflects the drilling pattern, spatial distribution and mine planning considerations. The Mineral Resource estimate was generated by ordinary kriging (OK) using Datamine software. The optimised pit shells were generated by SRK using the classified Mineral Resources. Various economic parameters such as mining and processing and G&A costs, gold and copper recovery, and pit slope angle were used in as input parameters for the resource pit shells. All open pit resources are stated above a 0.5 g/t gold equivalent cut-off.

This section describes the work undertaken by SRK and summarises the key assumptions and parameters used to prepare the Mineral Resource models.

Throughout the Mineral Resource estimate, the following abbreviations are used:

- Cu_proc – Cu grade, expressed as a percentage, also written as Cu%
- Au_ppm – Au grade, expressed as ppm or g/t

14.2 Drillhole Database

The drillholes used for the Kopsa MRE comprise 126 diamond drillholes for a total of 14,324 drilled metres. Of this, 12,381 drilled metres have been assayed for Cu% and Au g/t. This is summarised in Table 14-1 below.

Table 14-1: Available drillhole data

Number of drillholes	Total Drilled (m)	Assayed Metres (Cu%)	Assayed Metres (Au g/t)
126	14 324	12 381	12 381

14.3 Data Validation

Only diamond drillholes drilled by Belvedere and Glenmore Highland were used in the Mineral Resource estimate. As discussed previously, the location of the drillhole collars, and supporting QAQC data could not be located for the GTK and Outokumpu drillholes. These holes were imported into Datamine, and were used to help guide the geological interpretation, but the grade data was not used for the block modelling. All available data was validated through the production of histograms and scatterplots and the use of the Datamine drillhole validation tools. This resulted in a de-surveyed drillhole file, with all errors being removed. SRK considers that the data is of a sufficient quality for use in the subsequent Mineral Resource estimate.

14.4 Geological Modelling and Domaining

14.4.1 Introduction

Two wireframes were constructed for the Kopsa Cu-Au deposit, one for Cu and one Au. The modelled units consist of one main body striking roughly east-west and dipping towards the south. The wireframes were constructed on the basis of the drillhole database as a whole, including the historical drilling though, as already commented, the lower confidence, historical drillholes were not used in the grade interpolation process.

14.4.2 Geological Modelling and Block Model Creation

The geological modelling of the two zones was conducted in a combination of Leapfrog and Datamine Studio 3 software and comprised the following:

- importing the collar, survey, assay and geology data into both Leapfrog and Datamine to create a de-surveyed drillhole file;
- importing the topography data file and combining it with aster-data to cover a wider surface;
- the creation of mineralization wireframes;
- the creation of an empty block model coded by zone to distinguish the different geological domains identified (Figure 14-3, Figure 14-5, and Table 14-2); and

The modelled units are illustrated in Figure 14-1 and Figure 14-2, which show the drillhole distribution and solid wireframe created for the Cu-zone, and Au-zone, respectively.

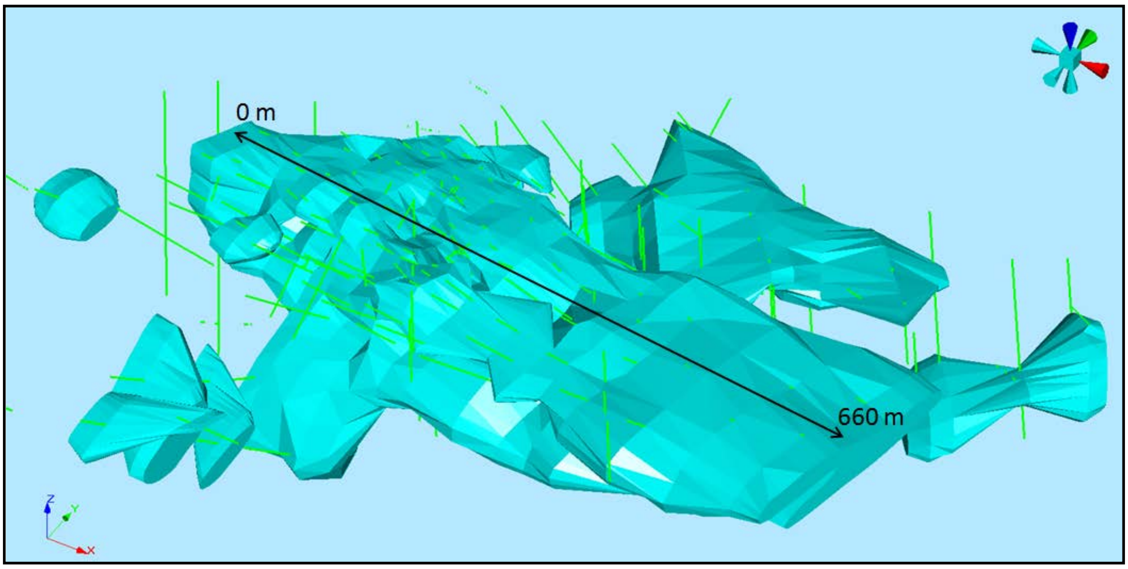


Figure 14-1: Wireframe for the Kopsa Cu-zone and drillhole locations (looking northwest) (Source: SRK, July 2013).

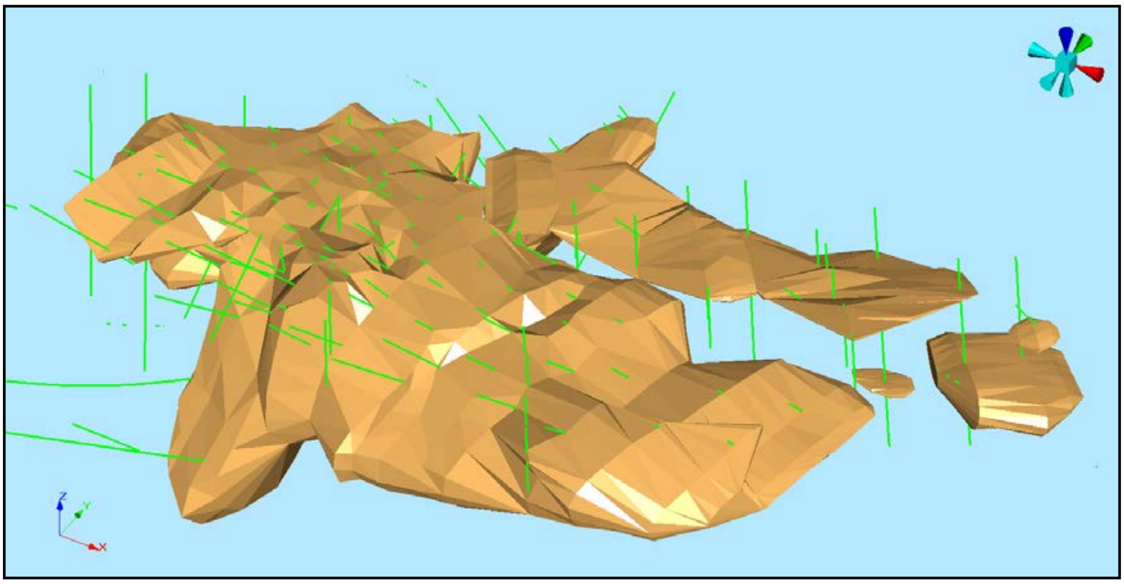


Figure 14-2: Wireframe for the Kopsa Au-zone and drillhole locations (looking northwest, see above figure for scale) (Source: SRK, July 2013).

Table 14-2 shows the coding applied to the various geological domains. The Cu and Au zone wireframe was used to code the model, with the area where the two wireframes overlap, called the combined zone (Zone 30). The block model coding is shown in Figure 14-3 to Figure 14-5.

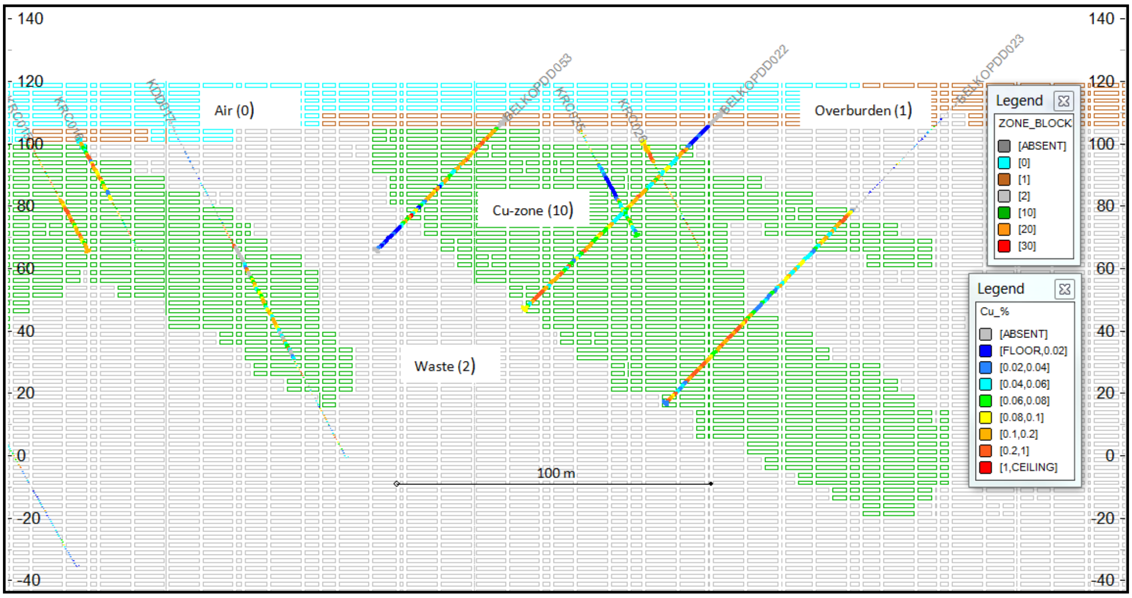


Figure 14-3: Block model (looking east). Drillholes coloured by Cu%. Block model coloured by zones, with Cu zone in green (Source: SRK, July 2013).

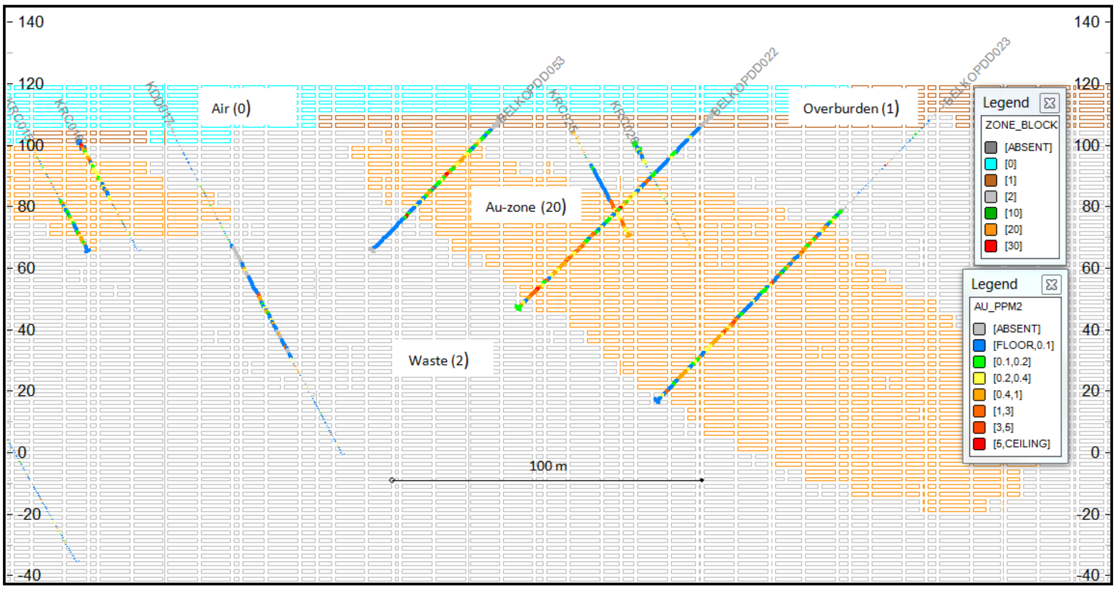


Figure 14-4: Block model (looking east). Drillholes coloured by Au_ppm. Block model coloured by zones, with Au zone in yellow (Source: SRK, July 2013).

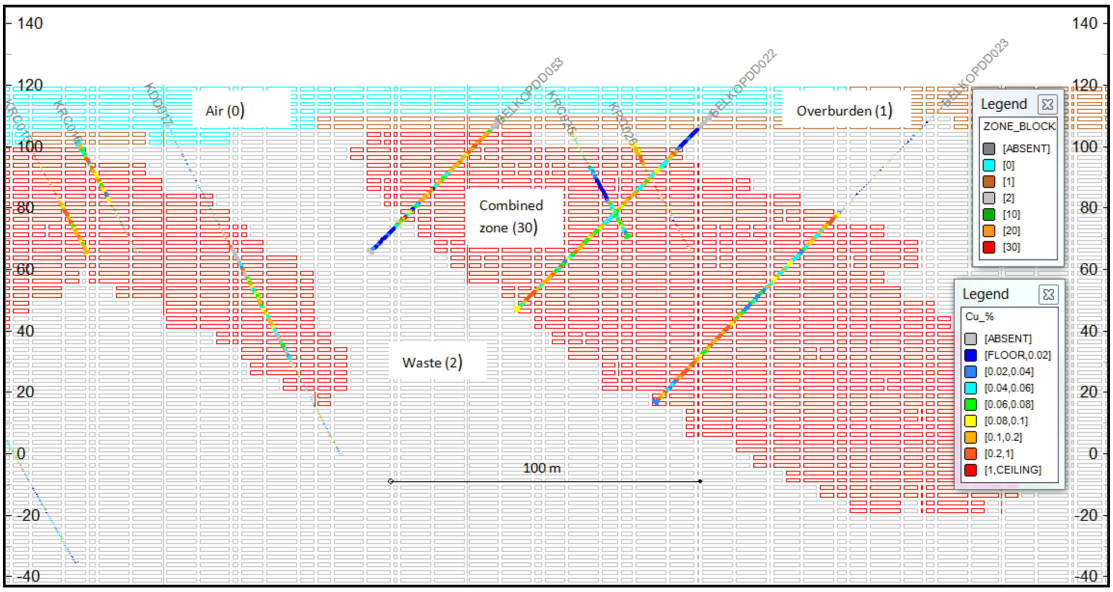


Figure 14-5: Block model (looking east). Drillholes coloured by Cu%. Block model coloured by zones, with combined (Cu+Au) zone in red (Source: SRK, July 2013).

Table 14-2: Zone codes created for Kopsa Copper Gold Project

Geology	Code
Air	0
Overburden	1
Waste	2
Cu zone	10
Au zone	20
Combined mineralization zone	30

14.5 Statistical Analysis of Raw Assay Data

Table 14-3 shows the statistics for the raw assay data, within each of the modelled domains. The mean Cu_proc grade in the Cu-zone is 0.15% and the mean Au_ppm grade in the Au-zone is 0.77 g/t.

The Coefficient of Variation (CoV) can be used to describe the shape of the distribution and is defined as the ratio of the standard deviation to the mean. A CoV greater than 1 indicates the presence of erratic high values that may have a significant impact on the final grade estimate. Table 14-3 shows that the majority the CoV values are within an acceptable range.

Table 14-3: Length weighted statistics for the Kopsa deposit

Variable	Unit	Zone	No. Samples	Min	Max	Range	Mean	Variance	Standard Deviation	CoV
CU_PROC	%	10	10,293	0.0037	2.28	2.28	0.15	0.02	0.13	0.89
AU_PPM	g/t	20	11,597	0.0005	48.4	48.4	0.77	3.85	1.96	2.56
AG_G/T	g/t	30	11,483	0.01	48.9	48.9	2.10	6.79	2.60	1.24
AS_G/T	g/t	30	12,238	1.00	200,000	199,999	3,738	5,823,007	7,199	1.93
S_PROC	%	30	11,374	0.01	9.25	9.24	0.69	0.23	0.48	0.70

14.6 Compositing

Data compositing is commonly undertaken to reduce the inherent variability that exists within the population and to generate samples more appropriate to the scale of the mining operation envisaged. It is also necessary for the estimation process, as all samples are assumed to be of equal support, and should therefore be of equal length.

The majority of samples in the Kopsa drillhole file are 1 m in length with smaller samples being present at the geological contacts. Figure 14-6 shows the sample length distribution in the raw dataset for Kopsa. All samples have been composited to 2 m as increasing the sample to a larger composite length has little impact on the variability of the database.

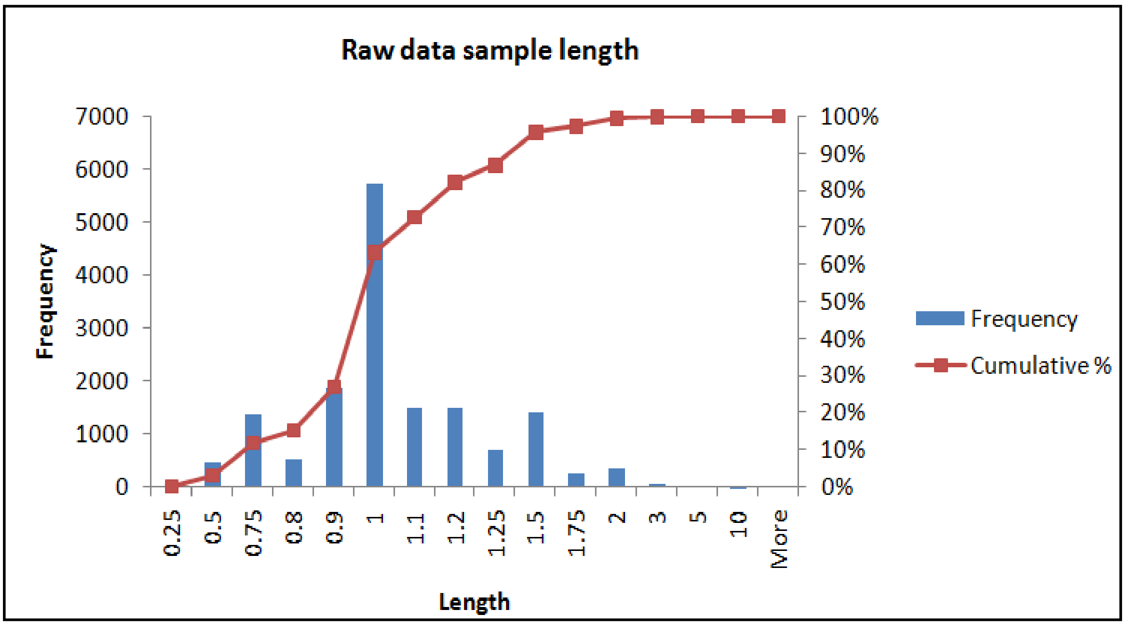


Figure 14-6: Raw data sample length (Source: SRK, July 2013).

14.7 Statistical Analysis of Composited Data

Statistics for the composited data, as restricted by the modelled wireframes are shown in Table 14-4. After compositing, the CoV has been lowered due to the reduced variability in the grades. This can also be seen in the histograms and log-histograms displayed in Figure 14-7, with near log-normal populations observed for Au within the Au zone and Cu within the Cu zone.

Table 14-4: 2 m composite statistics for Kopsa

Variable	Unit	Zone	No. Samples	Min	Max	Range	Mean	Variance	Standard Deviation	CoV
CU_PROC	%	10	3 031	0.01	1.42	1.42	0.15	0.01	0.11	0.7
AU_PPM	g/t	20	3 338	0.00	30.16	30.16	0.77	2.06	1.44	1.9
AG_G/T	g/t	30	3 151	0.01	22.54	22.53	2.06	4.16	2.04	1.0
AS_G/T	g/t	30	3 481	1.00	105 654	105 653	3 691	27 102 872	5 206	1.4
S_PROC	%	30	3 095	0.04	4.70	4.66	0.69	0.14	0.37	0.5

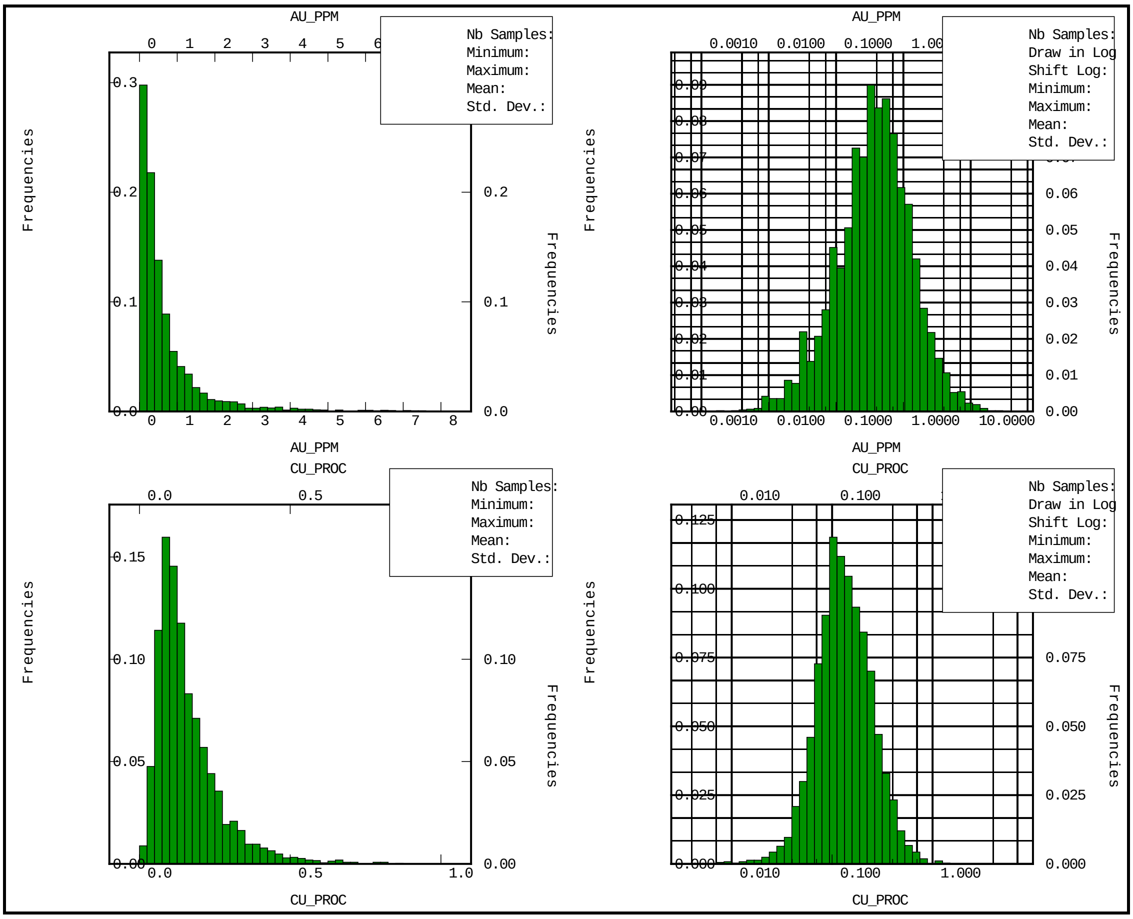


Figure 14-7: Histograms and log-histograms of composited drillholes (Source: SRK, July 2013).

14.8 Grade Capping

The statistical analysis of the composite data indicated that grade capping was not required since there were no extreme outliers encountered. Further, it was planned that any isolated higher grade samples would to some extent be accounted for in the kriging process as a minimum threshold was set in terms of the minimum number of composites needed to estimate a block value.

14.9 Density Analysis

Over 1,650 density measurements have been measured to date within the mineralised zone, with minimal variation, and an average bulk density of 2.73 g/cm^3. This value was applied to all blocks in the model due to the consistent lithology.

14.10 Geostatistical Analyses

14.10.1 Variography - Introduction

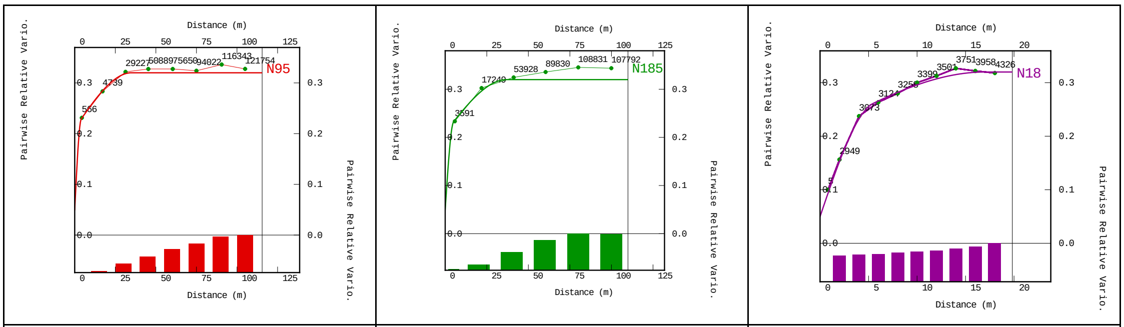
The composited drillhole file, as limited by the mineralization wireframe, was imported into Isatis software for the variographic analysis. Experimental directional pairwise semi-variograms were produced separately for Cu in the Cu-zone and Au in the Au-zone.

Down-hole semi-variograms were produced using a 2 m lag so as to allow the short-scale structures and nugget variance to be determined. Pairwise directional variograms were then produced with the nugget fixed from the down-hole variogram, and using a lag spacing of between 15 and 30 m (depending on the quality of the variogram) with a 50% tolerance being applied to the lag spacing. The rotation parameters used for the experimental variograms are consistent with the dip and strike of the domains, with a mean azimuth of 95° and a mean dip of 20°.

The Au estimation in the Cu-zone, in the parts of the zone that were not overlapping with the Au-zone, used the variogram parameters determined for the Cu data. Similarly the Cu estimation in the Au-zone, in the parts of the zone that were not overlapping with the Cu-zone, used the variogram parameters determined for the Au data.

14.10.2 Variography – Cu zone

The variography for the Cu-zone produced fairly robust directional pairwise semi-variograms for Cu, indicating that the drill spacing is sufficient at present to quantify the spatial characteristics of the domains. Omni-directional and directional variograms were attempted, but pairwise directional variograms gave more robust structures and ranges and therefore deemed more appropriate to use. The modelled directional pairwise semi-variograms are shown in Figure 14-8.



Along Strike	Down Dip	Down-hole

Figure 14-8: Modelled Variograms for Cu-zone (Source: SRK, July 2013).

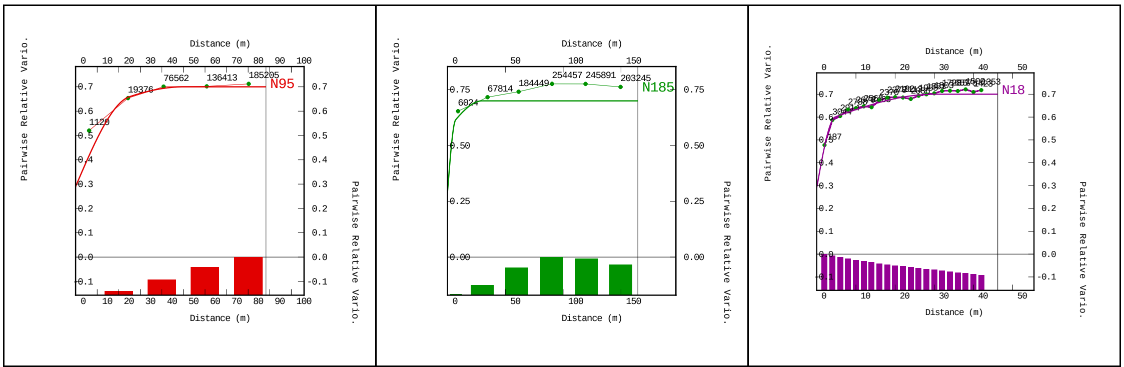
Table 14-5 shows the ranges, nugget effect and sills for the Cu-zone. The table also includes potential search ellipsoid radii as a result of the variography. SRK has used the maximum modelled range to derive potential search ellipsoid radii. Numerous other factors need to be considered in deriving an optimum search ellipsoid, and this is discussed further in Section 14.11.

Table 14-5: Variogram parameters for Cu-zone

	Modelled Variogram Parameters For Cu		
	Along Strike	**Down Dip**	**Down-hole**
Nugget Variance (Co)	0.05		
Nugget Effect (%)	**16%**		
1st Range (A1)	4	5	5.5
1st Sill (C1)	0.16		
2nd Range (A2)	35	37	17
2nd Sill (C3)	0.11		
Total Sill (Co + C)	0.32		
Search Ellipsoid Radii			
Total Range	35	37	17
Rounded	35	35	15
Potential Search Ellipsoid Radii	**35**	**35**	**15**

14.10.3 Variography – Au-zone

The variography for the Au-zone also produced fairly robust directional pairwise semi-variograms for Au, indicating that the drill spacing is sufficient at present to quantify the spatial characteristics of the domains. Omni-directional and directional variograms were attempted, but pairwise directional variograms gave more robust structures and ranges and therefore deemed more appropriate to use. The modelled directional pairwise semi-variograms are shown in Figure 14-9.



Along Strike	Down Dip	Down-hole

Figure 14-9: Modelled Variograms for Au-zone (Source: SRK, July 2013).

Table 14-6 shows the ranges, nugget effect and sills for the Au-zone. The table also include potential search ellipsoid radii as a result of the variography. SRK has used the maximum modelled range to derive potential search ellipsoid radii. Numerous other factors need to be considered in deriving an optimum search ellipsoid.

Table 14-6: Variogram parameters for the Au-zone

	Modelled Variogram Parameters for Au		
	Along Strike	**Down Dip**	**Down-hole**
Nugget Variance (Co)	0.29		
Nugget Effect (%)	**41%**		
1st Range (A1)	25	7	4.7
1st Sill (C1)	0.28		
2nd Range (A2)	50	30	30
2nd Sill (C3)	0.13		
Total Sill (Co + C)	0.70		
Search Ellipsoid Radii			
Total Range	50	30	30
Rounded	50	30	30
Potential Search Ellipsoid Radii	**50**	**30**	**30**

14.10.4 Summary

The pairwise directional experimental semi-variograms produced for the Cu- and the Au-zones for Kopsa allowed reasonable robust variogram models to be generated along strike, down-hole and down-dip.

The results of the variography were used in the interpolation to assign the appropriate weighting to the samples pairs being utilised to calculate the block model grade. The total ranges modelled have also been used to help define the preliminary optimum search parameters and the search ellipse dimensions used in the interpolation. Ideally, sample pairs that fall within the range of the variogram where a strong covariance exists between the sample pairs should be utilised if the data allows. Applying the total range of the variograms in the search ellipse dimensions forces the interpolation to use samples where covariance between samples exists. The preliminary search ellipse radii are shown in Table 14-7. As a result of the variography, ordinary kriging (OK) was deemed the most appropriate interpolation technique to be applied to Cu and Au.

Table 14-7: Ranges and Search Ellipses

Element	Parameter	Along Strike (m)	Down Dip (m)	Across Strike (m)
Cu	Average Total Range	35	37	17
	Preliminary Search Ellipse	50	50	15
Au	Average Total Range	50	30	30
	Preliminary Search Ellipse	50	30	30

14.11 Quantitative Kriging Neighbourhood Analysis (QKNA)

To better define the ideal search parameters used in the interpolation, Quantitative Kriging Neighbourhood Analysis (QKNA) was also undertaken on the data.

QKNA, as presented by Vann et al (2003), is used to refine the search parameters in the interpolation process to help ensure ‘conditional unbiasedness’ in the resulting estimates. ‘Conditional unbiasedness’ is defined by David (1977) as …on average, all blocks Z which are estimated to have a grade equal to Zo will have that grade. The criteria considered when evaluating a search area through QKNA, in order of priority, are (Vann et al 2003):

- the slope of regression of the ‘true’ block grade on the ‘estimated’ block grade;
- the weight of the mean for a simple kriging;
- the distribution of kriging weights, and proportion of negative weights; and
- the kriging variance.

Under the assumption that the variogram is valid, and the regression is linear, the regression between the ‘true’ and ‘estimated’ blocks can be calculated. The actual scatter plot can never be demonstrated, as the ‘true’ grades are never known, but the covariance between ‘true’ and ‘estimated’ blocks can be calculated. The slope of regression should be as close to one as possible, implying conditional unbiasedness. If the slope of regression equals one, the estimated block grade will approximately equate to the unknown ‘true’ block grades (Vann et al, 2003).

During OK, the sum of the kriging weights is equal to one. When Simple Kriging (SK) is used, the sum of kriging weights is not constrained to add up to one, with the remaining kriging weight being allocated to the mean grade of the input data. Therefore, not only the data within the search area is used to krige the block grade, but the mean grade of the input data also influences the final block grade. The kriging weight assigned to the input data mean grade is termed the weight of the mean. The weight of the mean of a SK is a good indication of the search area as it shows the influence of the Screen Effect. A sample is ‘screened’ if another sample lies between it and the point being estimated, causing the weight of the screened sample to be reduced. The Screen Effect is stronger when there are high levels of continuity denoted by the variogram. A high nugget effect (low continuity) will allow weights to be spread far from a block in order to reduce bias (Vann et al 2003). The weight of the mean for a SK demonstrates the strength of the Screen Effect the larger the weight of the mean, the weaker the Screen Effect will be. The general rule is that the weight of the mean should be as close to zero as possible. QKNA is a balancing act between maximising the slope of regression, and minimising the weight of the mean for a SK (Vann et al, 2003). The margins of an optimised search will contain samples with very small or slightly negative weights. Visual checks of the search area should be made in order to verify this. The proportion of negative weights in the search area should be less than 5% (Vann et al 2003).

QKNA provides a useful technique that uses mathematically sound tools to optimise a search area. It is an invaluable step in determining the correct search area for any estimation or simulation exercise.

In this case, neighbourhood tests were run separately on the Cu-zone and the Au-zone. In the first run, the search ellipsoid dimensions were fixed against the optimum ranges identified in the variography and as highlighted in Table 14-5 and Table 14-6. The search ellipsoid dimensions were then altered, and the number of blocks filled was noted. Using search parameters as defined from the variograms meant that there were parts of the block models which were un-estimated. During estimation, a number of statistics were written into the block model, including the slope of regression and kriging variance. Table 14-8 outlines the final chosen parameters, as defined from the QKNA tests, and used in the final block model estimation runs.

For the combined zone, the additional variables (Ag, As, and S) were interpolated into the model using IDW, as the variograms were not sufficiently clear for a robust model to be produced.

Table 14-8: Optimum model parameters, as defined by QKNA process

Zone	Search Ellipsoid Dimension (m)			Minimum Samples	Maximum Samples	Maximum Samples per Drillhole
	Along Strike	Down Dip	Across Strike			
Cu-zone	50	50	15	6	50	5
Au-zone	50	30	30	6	40	5
Combined-zone	50	50	30	6	50	5

During the QKNA process, each neighbourhood run was checked to ensure that an adequate number of blocks were filled ensuring that meaningful results were generated.

For the final chosen model, the distribution of Cu_proc slope of regression values is shown in Figure 14-10 and the distribution of Au_ppm slope of regression values is shown in Figure 14-11. A high slope of regression (>0.8) can be seen around well-informed blocks with the slope of regression value decreasing towards the base of the model where the blocks are less well-informed with sample data. The slope of regression data shows that the central and western portion of the deposit is better informed with data than the northern and northeast portion of the deposit.

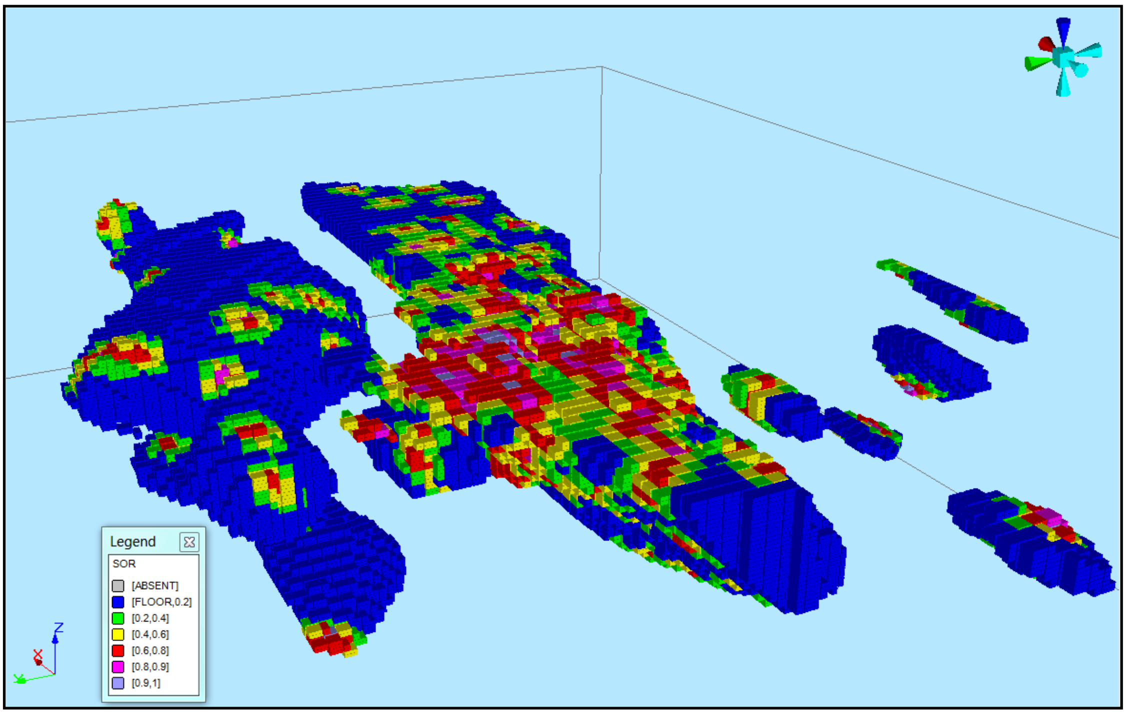


Figure 14-10: Kopsa block model coloured by slope of regression for Cu (looking ESE) (Source: SRK, July 2013).

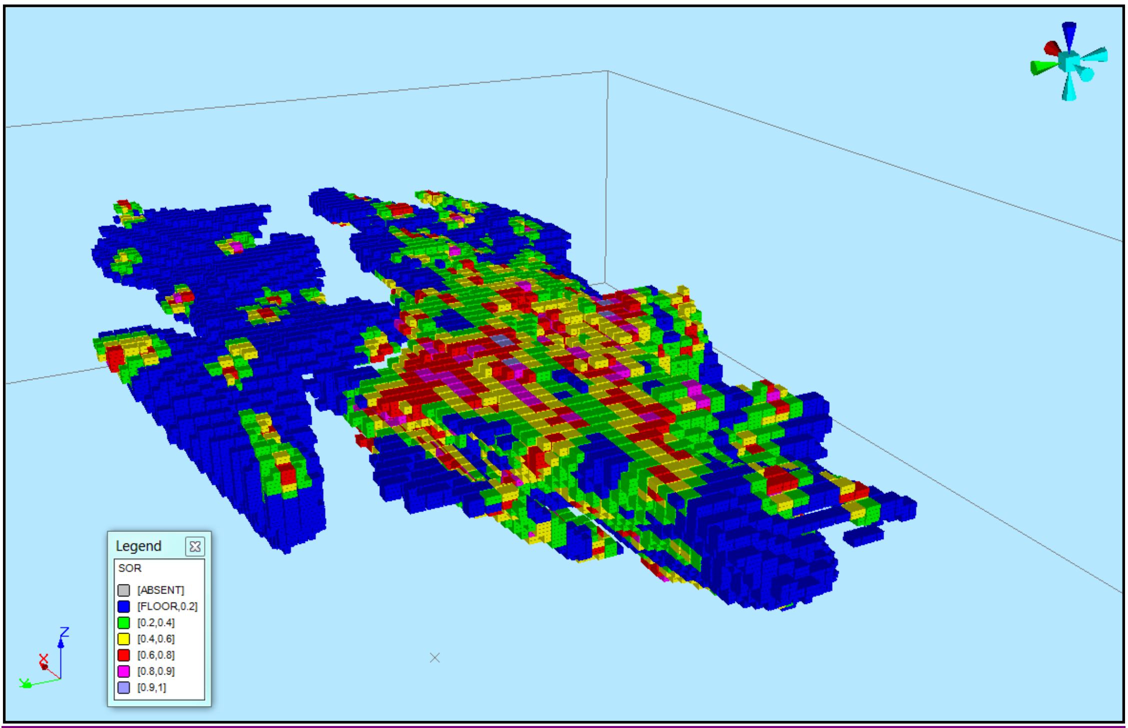


Figure 14-11: Kopsa block model coloured by slope of regression for Au (looking ESE) (Source: SRK, July 2013).

14.12 Block Modelling

14.12.1 Interpolation

An empty block model was generated with block dimensions as shown in Table 14-9, and coded using the grade shell wireframes. These block dimensions approximate half the drillhole spacing at Kopsa northeast. Due to the relatively low nugget effect observed for Kopsa, it is deemed appropriate to use blocks slightly smaller than half the drillhole spacing as it is assumed that blocks that are not supported by drillhole intersections are supported by data within the short scale range observed in the variograms. The results of the QKNA study also highlight that the blocks in the Kopsa southwest deposit are well supported by data. A block height of 5 m was chosen, being the assumed working bench height of the operating pit. Table 14-9 summarises the block model parameters.

Table 14-9: Block Model Framework

Coordinate	Origin	Block Size (m)	Number of Blocks
X	2560850	10	130
Y	7074700	10	90
Z	-160	5	56

Grades of CU_PROC and AU_PPM were interpolated into the model using OK and the kriging parameters given in Table 14-8. The parameters AG_G/T, AS_G/T and S_PROC were interpolated using Inverse Distance Weighting (IDW).

Cu was interpolated into the Cu-zone utilising the variography data determined for the Cu-zone (10). Au was interpolated into the Au-zone utilising the variography data determined for the Au-zone (20).

Combined zone of Cu-zone and Au-zone - Zone 30 - was used for interpolating background Au, Cu, Ag, As and S to the model using IDW. The kriged Cu and Au grades for the corresponding Cu- and Au-zones were overprinted on to the IDW grades, leaving IDW-interpolated Au grades where the Cu-zone did not overlap with the Au-zone, and vice versa.

14.12.2 Search Ellipse Parameters

The strike of the Kopsa deposit is near east-west and dips towards the south. Figure 14-12 shows the search ellipse generated for the Kopsa deposit, with the dip and strike of the ellipsoid corresponding with the dip and strike of the mineralization wireframes.

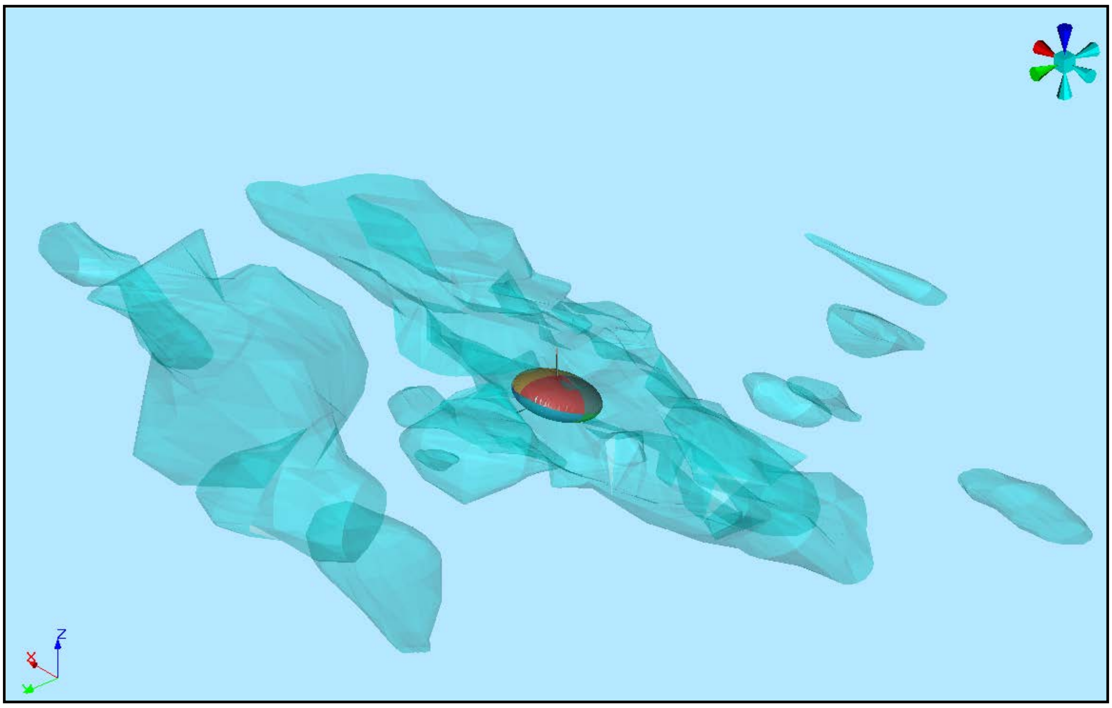


Figure 14-12: First pass search ellipses used in the interpolation of Kopsa (looking east southeast) (Source: SRK, July 2013).

Grades were interpolated in three separate runs. The first pass used the optimum parameters determined by the QKNA testing. The second run doubled the dimensions of the search ellipsoid, and the third run multiplied the original search ellipsoid by a factor of ten. The third run was designed to interpolate grades into any blocks not estimated in runs one and two. SRK notes that the confidence in the resulting grades is lower, as the search ellipsoid will have incorporated samples that are significantly outside the variogram range.

Table 14-10 to Table 14-12 illustrate the search ellipsoid parameters used for the three estimation runs respectively for the Cu-zone, the Au-zone and the combined zone.

Table 14-10: Search ellipse parameters for Cu

Zone	Dip Direction (°)	Dip (°)	Run	Along Strike Radii	Down Dip Radii	Across Strike Radii	Minimum Samples	Maximum Samples
Cu-zone	185	20	1	50	50	15	6	50
			2	100	100	30	6	50
			3	500	500	150	6	50

Table 14-11: Search ellipse parameters for Au

Zone	Dip Direction (°)	Dip (°)	Run	Along Strike Radii	Down Dip Radii	Across Strike Radii	Minimum Samples	Maximum Samples
Au-zone	185	20	1	50	30	30	6	40
			2	100	60	60	6	40
			3	500	300	300	6	40

Table 14-12: Search ellipse parameters for Combined zone

Zone	Dip Direction (°)	Dip (°)	Run	Along Strike Radii	Down Dip Radii	Across Strike Radii	Minimum Samples	Maximum Samples
Combined-zone	185	20	1	50	50	30	6	50
			2	100	100	60	6	50
			3	500	500	300	6	50

14.13 Block Model Validation

14.13.1 Introduction

The block model has been validated using the following techniques:

- visual inspection of block grades in plan and section and comparison with drillhole grades;
- comparison of global mean block grades and sample grades; and
- Validation plots.

14.13.2 Visual Validation

Figure 14-13 to Figure 14-16 show examples of the visual validation checks between block grades and the input composite grades for Cu% and Au_ppm. The grade distribution pattern follows, showing that the search ellipsoid orientation has been used appropriately.

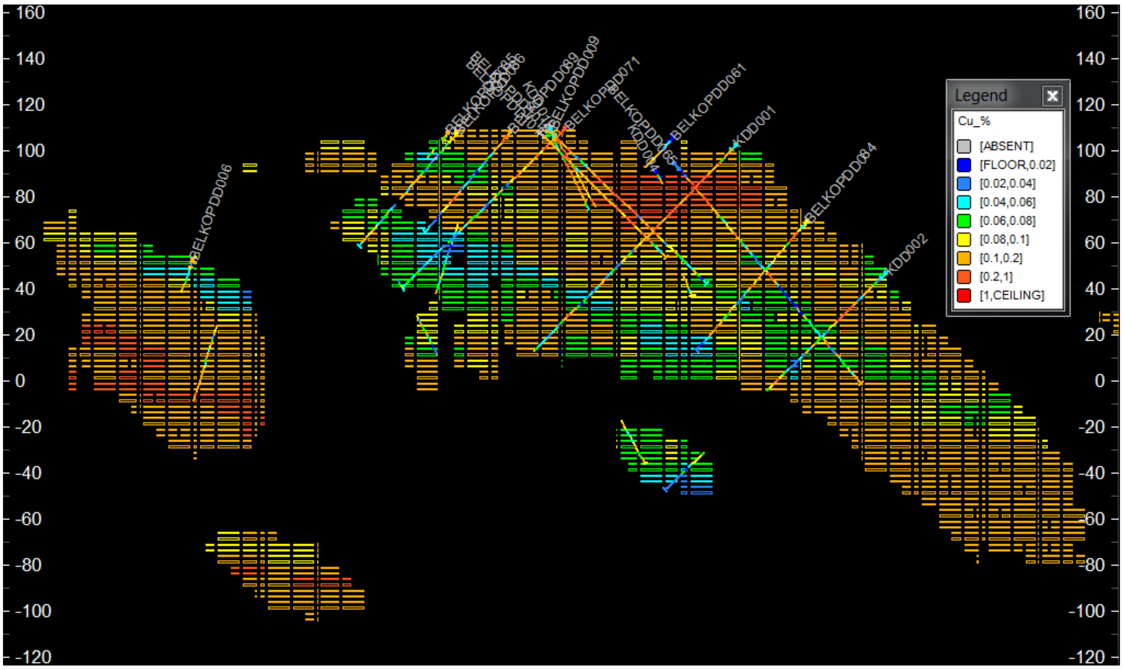


Figure 14-13: Visual validation of Cu block grades against 2 m composite sample grades for Kopsa, cross section in west (looking east) (Source: SRK, July 2013).

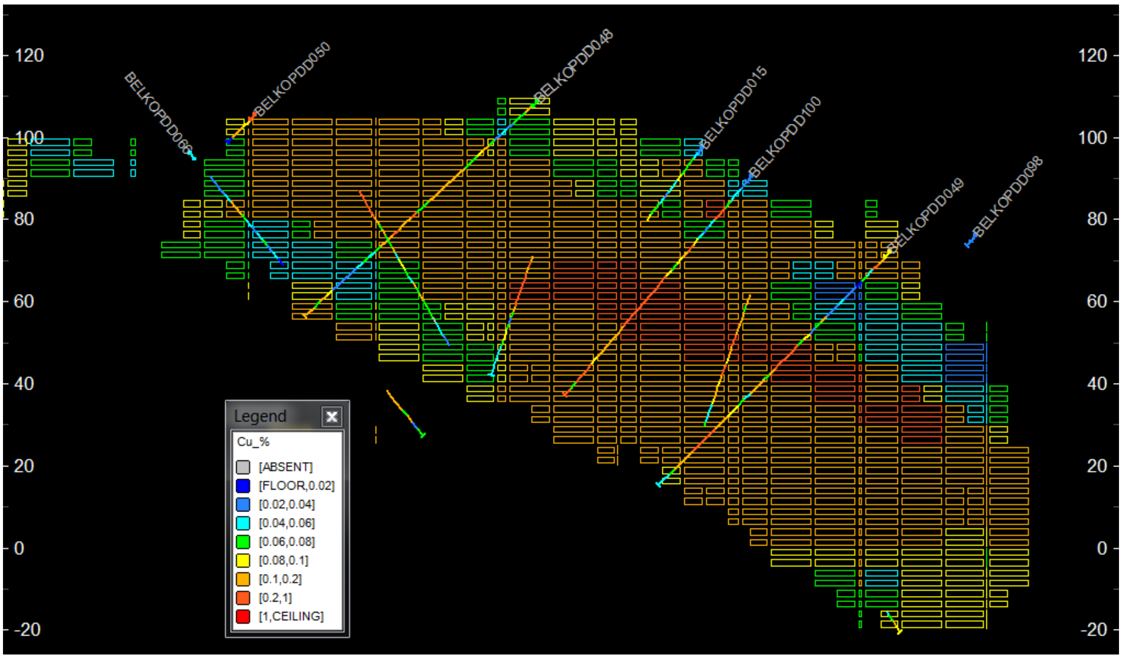


Figure 14-14: Visual validation of Cu block grades against 2 m composite sample grades for central parts of Kopsa, cross section looking east (Source: SRK, July 2013).

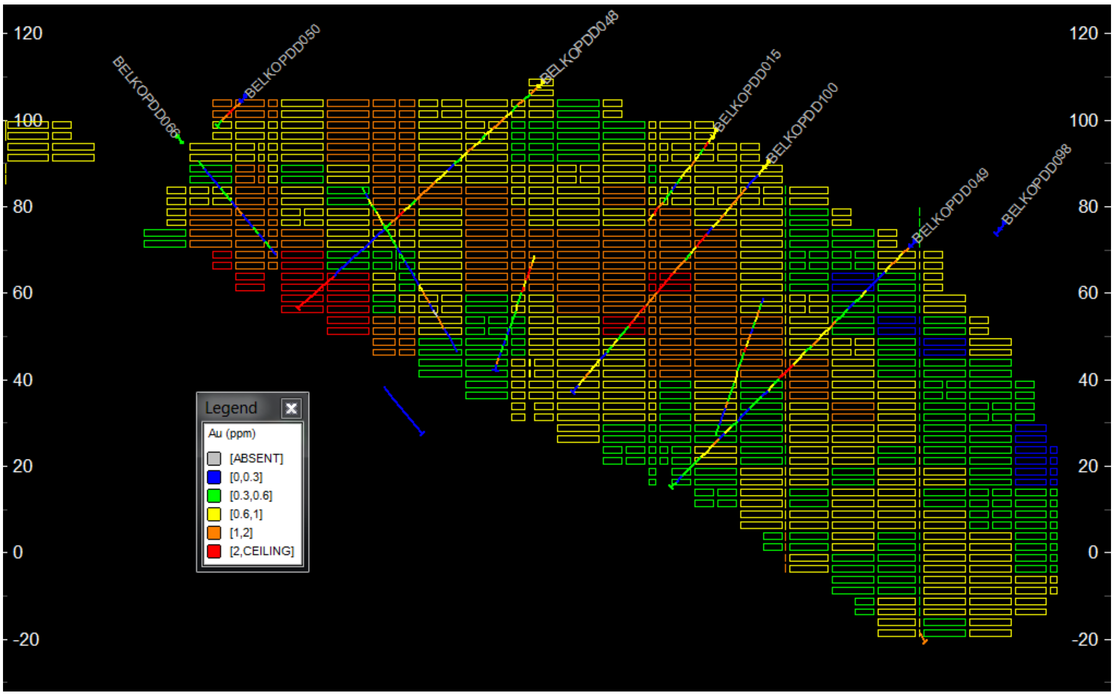


Figure 14-15: Visual validation of Au block grades against 2 m composite sample grades for central parts of Kopsa, cross section (looking east) (Source: SRK, July 2013).

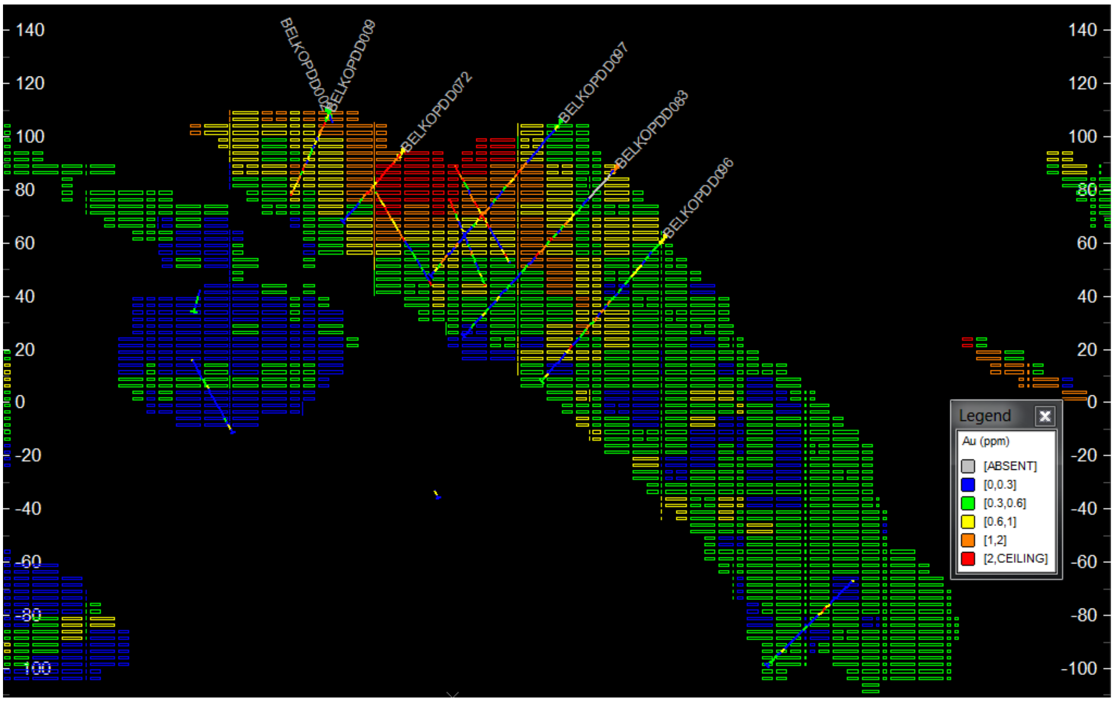


Figure 14-16: Visual validation of Au block grades against 2 m composite sample grades for western parts of Kopsa, cross section (looking east) (Source: SRK, July 2013).

Table 14-13 shows a comparison of the global block mean grades with the global sample means grades for Cu%, Au_ppm, Ag_g/t, As_g/t and S% within the combined zone and additionally for Au_ppm in the Au-zone and for Cu% in the Cu-zone.

Overall, SRK is confident that the interpolated grades are a reasonable reflection of the available sample data with the key grade fields being well within acceptable limits.

Table 14-13: Comparison of block and sample mean grades

Zone	Zone Code	Variable	Unit	Block Model Mean Grade	Composite Mean Grade	Actual difference	% Difference
Cu-zone	10	CU_PROC	%	0.15	0.15	0.00	1.5
Au-zone	20	AU_PPM	g/t	0.65	0.77	0.12	18.1
Combined	30	AG_G/T	g/t	2.25	2.06	-0.19	8.5
Combined	30	AS_G/T	g/t	3773	3691	-82.00	2.2
Combined	30	S_PROC	%	0.68	0.69	0.01	1.2

14.13.3 Validation Plots

As part of the validation process, the block model and input samples that fall within defined sectional or elevation criteria were compared and the results displayed graphically to check for visual discrepancies between grades.

Whilst this process does not truly replicate the samples used in the estimation of each block, the process of sectional validation quickly highlights areas of concern within the model and enables a more thorough and quantifiable check to be undertaken in specific areas of the model. Each graph also shows the number of samples available for the estimation. This provides information relating to the support of the blocks in the model. Only those blocks estimated within search volume one were compared, as this represents the estimated data using the optimum sample criteria.

Figure 14-17 and Figure 14-18 show the Cu and Au validation slices through the deposit. They show generally moderate to good correlation to the sample data, with a smoothing effect on the large outliers.

SRK is confident that the block model grades are a reasonable reflection of the composite sample grades.

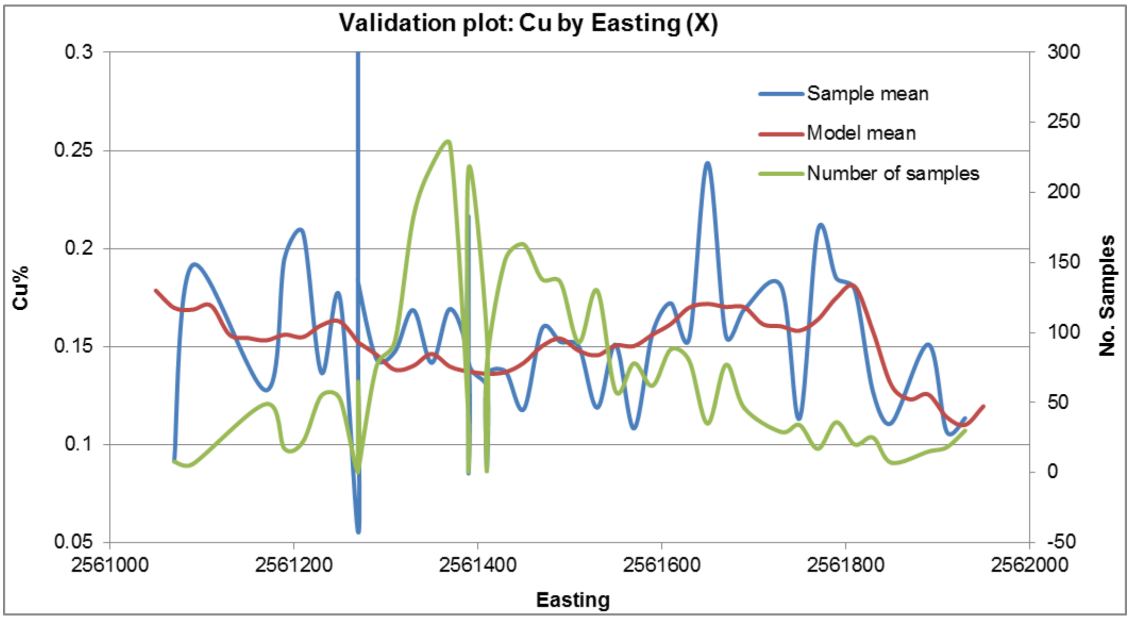


Figure 14-17: Validation plot by Easting (X) for Cu within Cu Zone (Source: SRK, July 2013).

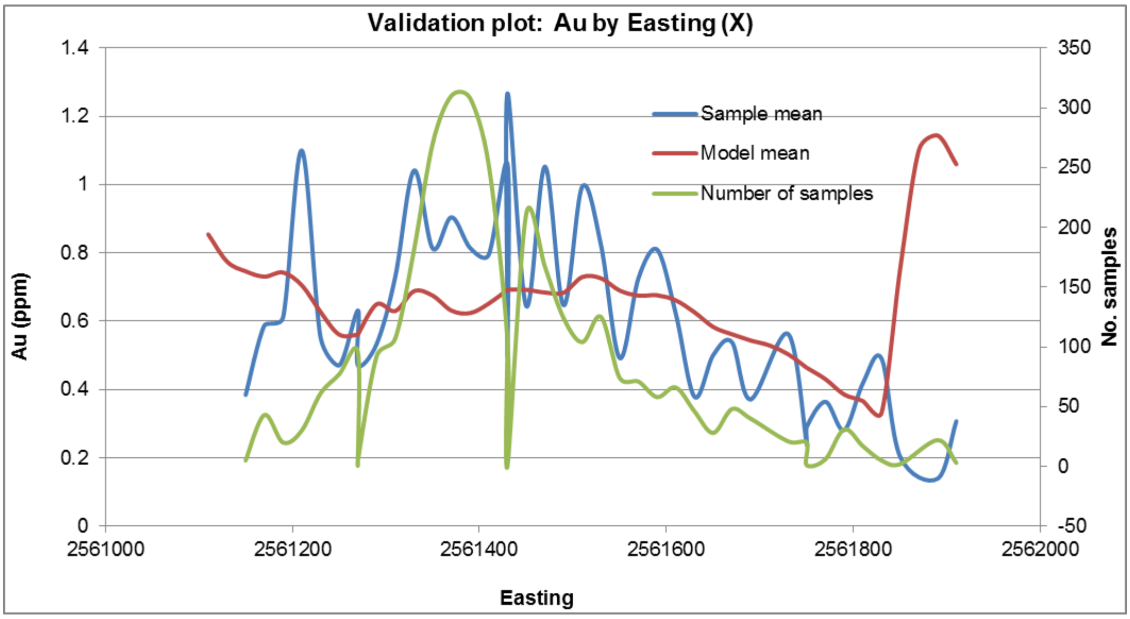


Figure 14-18: Validation plot by Easting (X) for Au within Au Zone (Source: SRK, July 2013).

14.14 Mineral Resource Classification

14.14.1 CIM Definitions

The definitions given in the following section are taken from the 2000 Canadian Institute of Mining Standing Committee on Reserve Definitions' guidelines on Mineral Resources and Reserves, and comply with the requirements of National Instrument 43-101.

Mineral Resource

Mineral Resources are sub-divided, in order of increasing geological confidence, into Inferred, Indicated and Measured categories. An Inferred Mineral Resource has a lower level of confidence than that applied to an Indicated Mineral Resource. An Indicated Mineral Resource has a higher level of confidence than an Inferred Mineral Resource but has a lower level of confidence than a Measured Mineral Resource.

A Mineral Resource is a concentration or occurrence of natural, solid, inorganic or fossilised organic material in or on the Earth's crust in such form and quantity and of such a grade or quality that it has reasonable prospects for economic extraction. The location, quantity, grade, geological characteristics and continuity of a Mineral Resource are known, estimated or interpreted from specific geological evidence and knowledge.

The term Mineral Resource covers mineralization and natural material of intrinsic economic interest which has been identified and estimated through exploration and sampling and within which Mineral Reserves may subsequently be defined by the consideration and application of technical, economic, legal, environmental, socio-economic and governmental factors. The phrase 'reasonable prospects for economic extraction' implies a judgement by the Qualified Person in respect of the technical and economic factors likely to influence the prospect of economic extraction. A Mineral Resource is an inventory of mineralization that, under realistically assumed and justifiable technical and economic conditions, might become economically extractable. These assumptions must be presented explicitly in both public and technical reports.

Inferred Mineral Resource

An ‘Inferred Mineral Resource’ is that part of a Mineral Resource for which quantity and grade or quality can be estimated on the basis of geological evidence and limited sampling and reasonably assumed, but not verified, geological and grade continuity. The estimate is based on limited information and sampling gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes.

Due to the uncertainty which may attach to Inferred Mineral Resources, it cannot be assumed that all or any part of an Inferred Mineral Resource will be upgraded to an Indicated or Measured Mineral Resource as a result of continued exploration. Confidence in the estimate is insufficient to allow the meaningful application of technical and economic parameters or to enable an evaluation of economic viability worthy of public disclosure. Inferred Mineral Resources must be excluded from estimates forming the basis of feasibility or other economic studies.

Indicated Mineral Resource

An ‘Indicated Mineral Resource’ is that part of a Mineral Resource for which quantity, grade or quality, densities, shape and physical characteristics can be estimated with a level of confidence sufficient to allow the appropriate application of technical and economic parameters, to support mine planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough for geological and grade continuity to be reasonably assumed.

Mineralization may be classified as an Indicated Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such as to allow confident interpretation of the geological framework and to reasonably assume the continuity of mineralization. The Qualified Person must recognise the importance of the Indicated Mineral Resource category to the advancement of the feasibility of the project. An Indicated Mineral Resource estimate is of sufficient quality to support a Preliminary Feasibility Study which can serve as the basis for major development decisions.

Measured Mineral Resource

A ‘Measured Mineral Resource’ is that part of a Mineral Resource for which quantity, grade or quality, densities, shape, physical characteristics are so well established that they can be estimated with confidence sufficient to allow the appropriate application of technical and economic parameters, to support production planning and evaluation of the economic viability of the deposit. The estimate is based on detailed and reliable exploration, sampling and testing information gathered through appropriate techniques from locations such as outcrops, trenches, pits, workings and drillholes that are spaced closely enough to confirm both geological and grade continuity.

Mineralization or other natural material of economic interest may be classified as a Measured Mineral Resource by the Qualified Person when the nature, quality, quantity and distribution of data are such that the tonnage and grade of the mineralization can be estimated to within close limits and that variation from the estimate would not significantly affect potential economic viability. This category requires a high level of confidence in, and understanding of, the geology and controls of the mineral deposit.

14.14.2 Kopsa Classification

14.14.2.1 Introduction

To classify the Kopsa deposit, the following key indicators were used:

- Geological complexity;
- Quality of data used in the estimation;
- QAQC, density analysis;
- Results of the geostatistical (variography) analysis;
- QKNA results; and
- Quality of the estimated block model.

14.14.2.2 Geological Complexity

Due to the extensive, close spaced drilling, SRK considers that the geological continuity between sections is well understood and that the current geological interpretation is well supported. The mineralization has been modelled as two separate grade shells, based on Cu and Au. A statistical study of the Kopsa data shows a very low variability in the grade distribution with log-normal populations of data being present. SRK considers that the associated risk relating to geological complexity is low.

14.14.2.3 Quality of the Data used in the Estimation

The drilling programmes are supported by an extensive QAQC process, which included the insertion of CRMs, blanks, laboratory duplicates and the use of an umpire laboratory. Overall SRK is confident that the results of the QAQC analysis have validated the accuracy of the database being used to generate the Mineral Resource Estimate.

A dataset of density has also been generated by the Company and an average bulk density has been calculated. SRK has used that average density of 2.73 g/cm^3 for the estimate. The deposit consists of a single lithology (tonalite), which has a generally consistent density. SRK considers that the tonnages estimated for the Kopsa deposit are reasonable.

14.14.2.4 Statistical and Geostatistical Analysis

Geostatistical analysis of the composited assay data resulted in robust variogram models being produced for the deposit. This enabled the nugget and short-scale variation in grade to be determined with a high level of confidence. The detailed variography allowed for the determination of appropriate search ellipse parameters to be determined through the application of multiple QKNA tests prior to the grade interpolation.

14.14.2.5 Quality of the Estimated Block Model

SRK has validated the models using both visual and statistical methods. SRK is confident that the block models reflect the input data on both local and global scales.

14.14.2.6 Classification

Given the above, the Kopsa deposit has been classified into a combination of Measured, Indicated and Inferred categories.

Measured Resources at Kopsa have been assigned where the following criteria have been met:

- Low geological and grade complexity;
- Average drillhole spacing approximately equal to or less than the modelled geostatistical range; and
- Majority of blocks being estimated by the first pass search volume, using the optimum search parameters determined.

Indicated Resources at Kopsa have been assigned where the following criteria have been met:

- Low geological and grade complexity; and
- Majority of blocks being estimated by the first pass search volume, using the optimum search parameters determined.

Inferred Resources at Kopsa have been assigned where most blocks were estimated in the second pass search volume, and where drilling is noticeably wider. Blocks not meeting these criteria were not included in the Mineral Resource estimate. This material is mainly at depth, and in areas where the drilling is generally quite wide.

The above criteria have been used to model 3-D geometric shapes for each of the classification categories rather than to assign the classification on a block by block basis. Once the shapes had been produced, these were used to code the block model, and reviewed to ensure that the coded model reflected the understanding of the geological and grade continuity.

Figure 14-19 shows the block model coloured by classification.

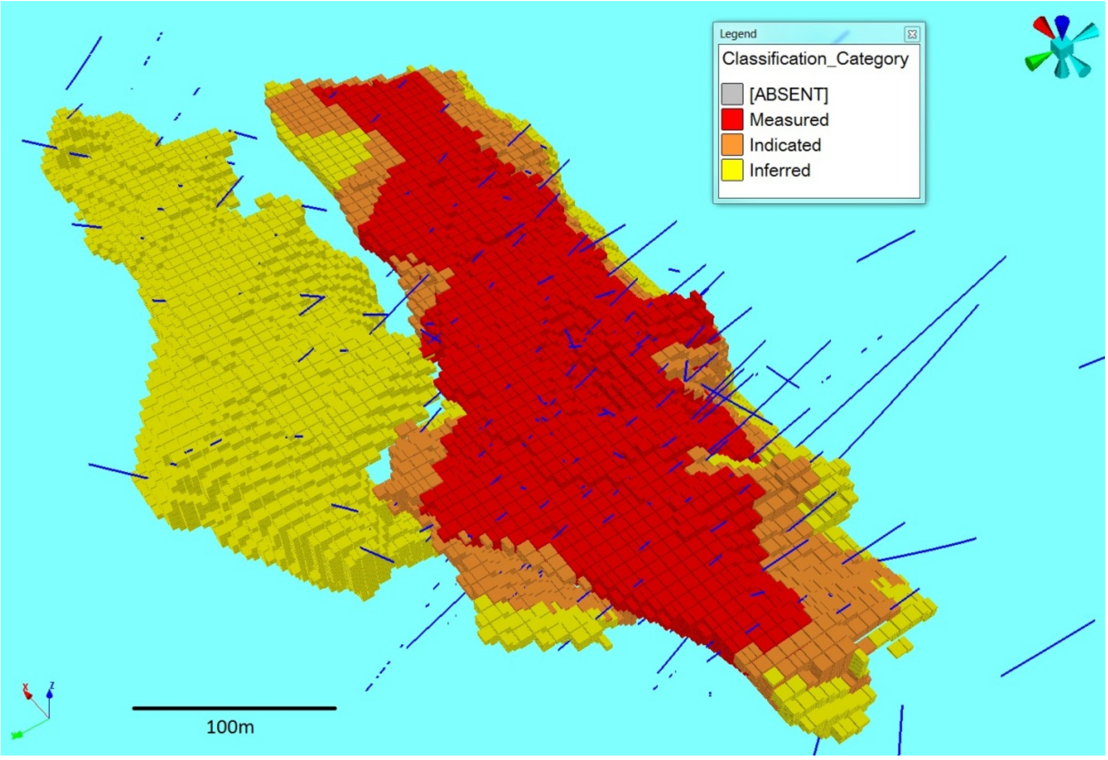


Figure 14-19: Kopsa classification. Red = Measured; Orange = Indicated; Yellow = Inferred; drillholes in green (looking ESE) (Source: SRK, July 2013).

14.15 Pit Optimisation for Mineral Resource Estimation

In order to derive the final Mineral Resource Statement, and so as to comply with the requirement that the resulting Mineral Resource must have reasonable prospects of economic extraction, the resulting blocks have been subjected to a Whittle pit optimisation exercise.

The optimisation requires the input of reasonable processing and mining cost parameters in addition to appropriate pit slope angles and processing recoveries.

Table 14-14 shows the assumptions applied in the Whittle optimisation.

The Whittle optimisation has assumed that the combined Au- and Cu- grade shell (Zone 30) are to be treated as the key potential material type.

Table 14-14: Whittle parameters

Geotechnical Parameters	
Overall Slope Angles FW/HW	45/55º
Metal Selling Prices	
Copper Price	7865 USD/t
Gold Price	1508 USD/oz
Mining Cost Factors	
Total Open Pit Mining Cost (Base RL)	3.5 USD/t
Base RL for optimisation	110 m
Incremental Mining Cost below BRL	0.05 USD/t/10м
Processing Cost Factors (includes G&A)	
Crushing, Grinding and Flotation	12.0 USD/t
Cyanidation	1.0 USD/t
Other Cost Factors	
Distance to Process Plant	20 km
Transport Cost	0.28 USD/t/km
Royalties	0% (Belvedere planning to purchase land)
Mining Parameters	
Mining Recovery	97%
Mining Dilution	5%
Production Capacity	1.0 Mtpa
Minimum Operating Width	35 m
Processing Parameters	
Processing Capacity	1.0 Mtpa
Recovery Cu	76.0%
Recovery Au	80.1%
Concentrate Grade Cu	22.5%

14.16 Gold Equivalent Calculation

Each block is assigned a gold equivalent (AuEq) based on the interpolated CU_PROC and AU_PPM in each block as well as using long term metal prices and assumed recoveries, as described above. The following calculation was used to assign AuEq values to each block:

$$AuEq\ (g/t) = 0.982830*Au\ (g/t) + 1.672011*Cu\ (\%)$$

14.17 Mineral Resource Statement

The Mineral Resource statement generated by SRK has been restricted to that material falling within the Whittle shell and above a cut-off grade of 0.50 g/t AuEq, representing the calculated marginal cut-off grade for the deposit. A USD7870 / t copper price, and USD1508 / Oz Au price, were used for the optimisation, which includes a 30% premium above the consensus long-term price, so as to include material with the potential to be extracted in the future not just that material that justifies extraction now, determined from over 30 market forecasts. SRK consider that the material included within the Whittle shell and above the cut-off grade demonstrates reasonable prospects for eventual economic extraction, as required by NI43-101 reporting standard.

Table 14-15 shows the resulting Mineral Resource Statement for Cu and Au for Kopsa.

The statement has been classified by Lucy Roberts (MAusIMM(CP)) in accordance with the CIM Definitions. The effective date for these statements is 02 October 2013.

Mineral resources that are not mineral reserves do not have demonstrated economic viability. Notwithstanding this, neither SRK nor the Company are aware of any factors (environmental, permitting, legal, title, taxation, socio-economic, marketing, political or other relevant factors) that could materially affect the potential of these to be exploited.

Table 14-15: Mineral Resource Statement (reported above a marginal cut-off grade of 0.5 g/t AuEq and within the Whittle shell)

Category	Tonnes (Mt)	Au (g/t)	Cu (%)	AuEq (g/t)	Ag (g/t)
Measured	11.5	0.83	0.15	1.07	2.17
Indicated	2.2	0.70	0.15	0.95	2.08
Measured+Indicated	**13.6**	**0.81**	**0.15**	**1.05**	**2.15**
Inferred	2.7	0.8	0.2	1.1	2.57

In total, the Kopsa deposit has been estimated to contain a Measured Mineral Resource of 11.5 Mt with mean grades of 0.83 g/t Au and 0.15% Cu, and an Indicated Mineral Resource of 2.2 Mt with mean grades of 0.70 g/t Au and 0.15% Cu. In addition to the Measured and Indicated Mineral Resources, SRK has derived an Inferred Mineral Resource estimate of some 2.7 Mt with mean grades of 0.8 g/t Au and 0.2% Cu. See Section 14.15 for parameters used in the process.

14.18 Grade Tonnage Curves

A grade-tonnage curve for Cu% is shown in Figure 14-20. The curve shows the relationship between the modelled tonnage and grade at increasing Cu % cut-offs and notably shows a rather steep decreasing tonnage with an associated increasing Cu% grade from a Cu% cut off of approximately 0.1% Cu.

A grade-tonnage curve for Au ppm is shown in Figure 14-21. The curve shows the relationship between the modelled tonnage and grade at increasing Au ppm cut-offs and notably shows a rather steep decreasing tonnage with an associated increasing Au ppm grade from an Au ppm cut off of approximately 0.3 ppm Au.

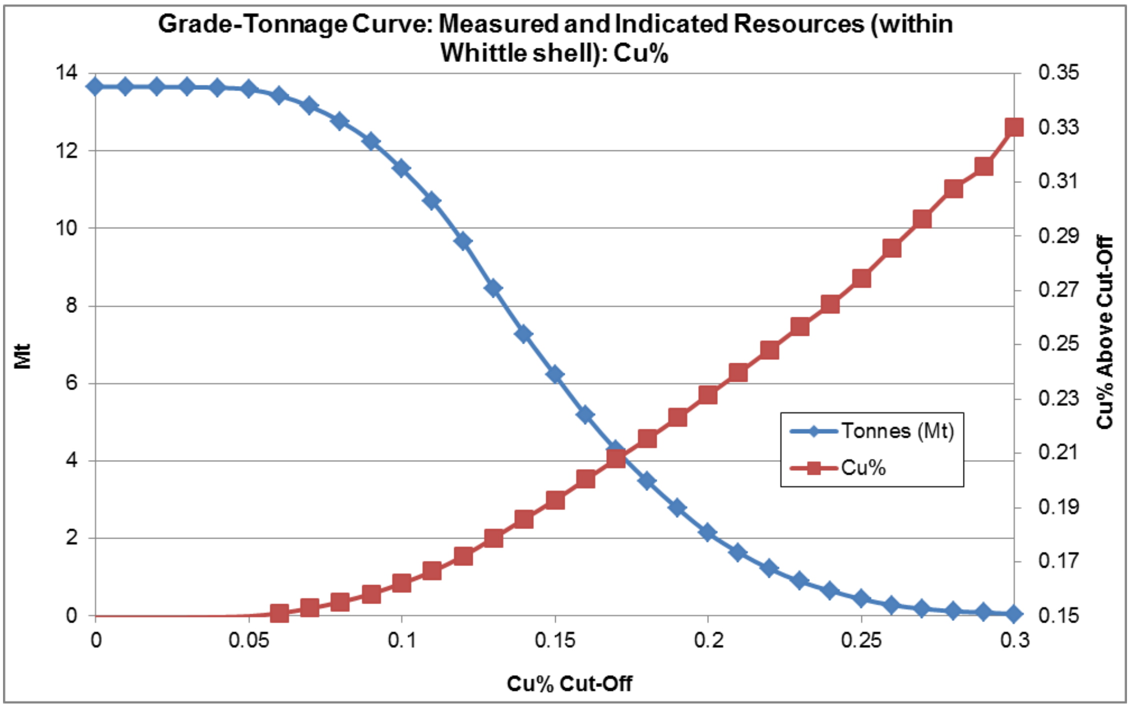


Figure 14-20: Kopsa Grade Tonnage Curve for Cu – Measured and Indicated Resources above Resource pit shell and above 0.5 AuEq cut-off (Source: SRK, July 2013).

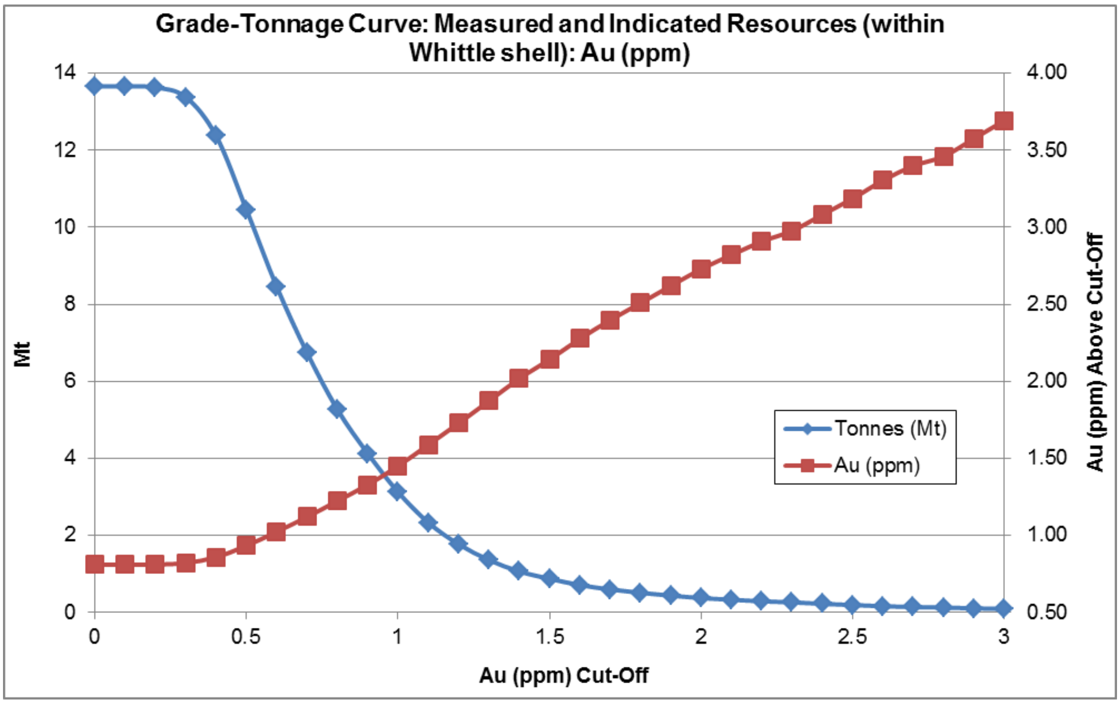


Figure 14-21: Kopsa Grade Tonnage Curve for Au – Measured and Indicated Resources above Resource pit shell and above 0.5 AuEq cut-off (Source: SRK, July 2013).

Table 14-16: Cu cut-off grade-tonnage results (Measured and Indicated)

Cu (%) Cut-Off	Kt	Cu (%)
0	15,489	0.15
0.05	15,316	0.15
0.10	12,743	0.16
0.15	6,889	0.19
0.20	2,393	0.23
0.25	488	0.27

Table 14-17: Au cut-off grade-tonnage results (Measured and Indicated)

Au (g/t) Cut-Off	Kt	Au (g/t)
0.05	15,482	0.81
0.10	15,442	0.81
0.20	15,262	0.82
0.30	14,539	0.85
0.40	13,365	0.89
0.50	11,806	0.95
0.60	9,916	1.03
0.70	7,904	1.12
0.80	6,067	1.23
0.90	4,680	1.35
1.00	3,534	1.48
1.10	2,699	1.61
1.20	2,081	1.75
1.30	1,556	1.92
1.40	1,253	2.06
1.50	1,057	2.18

14.19 Comparison to 2012 Outotec MRE

The previous Mineral Resource estimate was prepared by Pekka Lovén, Outotec Oy, who is a Qualified Person (QP) as defined by Canadian NI 43-101 regulations; the estimate has an effective date of 29 October 2012 and is reproduced in Table 14-18.

Table 14-18: Resource Statement by Outotec of 29 October 2012 above a cut-off grade of 0.4 g/t Au

Category	Tonnes (Kt)	Au g/t	Cu ppm	As ppm
Measured	-	-	-	-
Indicated	6 680	1.04	1 526	4 886
Meas+Ind	**6 680**	**1.04**	**1 526**	**4 886**
Inferred	1 800	0.76	1 761	5 191

The key changes between the 2012 and 2013 SRK Mineral Resource statements are the new grade shells created for both the Au- and the Cu-zones, resulting from re-interpretation carried out by SRK. In addition, Outotec did not include the northern area of mineralization, considered potentially economic by SRK. This has resulted in increased tonnage in the SRK resource model. A large area in the centre of the deposit is well-drilled on a tight grid, and provided statistically robust block estimates, which SRK has classified as a Measured Resource. The 2012 MRE did not include any Measured Resources. In addition, the Outotec MRE utilised a considerably higher cut-off grade of 0.4 g/t Au to report resources.

14.20 Exploration Potential

SRK notes that there is potential for increasing the resource tonnage, along with upgrading the current classification of existing estimated mineralisation. The following exploration potential is noted:

- There is potential to extend the current resource with further exploration drilling mainly down-dip and along strike to the east (as shown in Figure 14-22) and in the north of the deposit (open down-dip and along-strike).
- Infill drilling in areas of sparse drilling data would likely result in upgrading of resource categories, particularly in the northern area where large areas of Inferred Resources have been outlined.
- Currently identified high-grade mineralised zones, both in northeast and in west areas should be investigated further with additional drilling. Figure 14-22 shows the block model coloured by AuEq grade, with high-grade areas shown in the northeast and central-west areas.

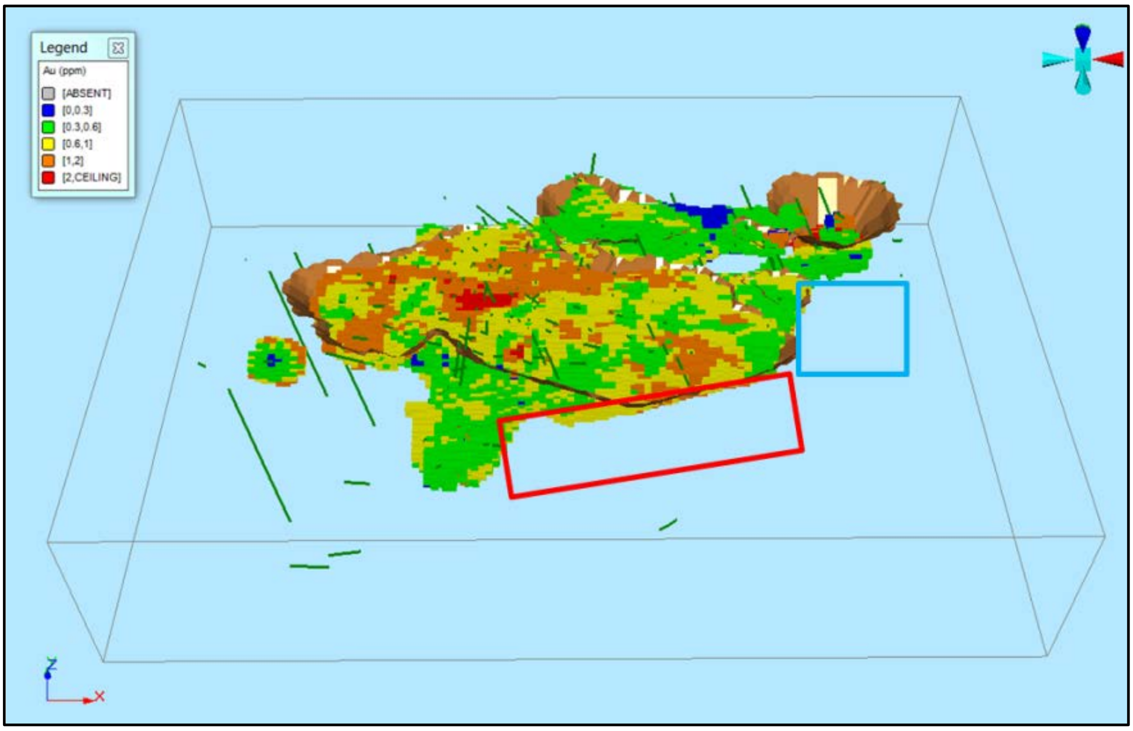


Figure 14-22: Kopsa pit shell with blockmodel coloured by Au_Eq, showing down-dip (red box) and along strike (blue box) exploraiton potential (Source: SRK, September 2013).

15 MINERAL RESERVE ESTIMATES

Not applicable.

16 MINING METHODS

16.1 Overview

This section presents the hydrological, geotechnical and mining inputs into the design of the Kopsa open pit, and the resulting mining schedules.

SRK has evaluated the potential to mine the deposit using an open pit mining method and reviewed the available geotechnical and hydrogeological information to determine suitable slope angles. Commercial pit optimisation software was then applied to the geological block model to determine the potential optimal pit boundary for economic analysis. SRK has also produced a preliminary production schedule and estimated the mining costs.

A conventional approach to open pit mining using an excavator-truck configuration is proposed for mining. A production rate of 1.2 Mtpa is considered appropriate by SRK based on current mining and metallurgical process assumptions and certain environmental limitations. SRK has considered an owner-operator approach for all mining and transport to processing facilities which are located approximately 20 km via sealed road from the deposit.

16.2 Geotechnical Analysis

SRK has reviewed the following sources of information as part of the geotechnical assessment:

- A regional structural geology report prepared by the Geological Survey of Finland;
- Fracture zones identified and described in this report;
- Structural similarities, and therefore comparisons with, to the Björkdal mine in Skellefteå.
- An interpretation of the outcropping structures at Kopsa which appear to conform to the structural analysis based on drill-hole data from Kopsa; and
- A summary of Kopsa Structures as given in a Company Internal report.

The importance of structures was identified in the early stages of the project and efforts were made to collect structural data early as possible, including oriented core and outcrop mapping.

Stereographic analysis has been performed on mineralised quartz veins and natural rock fractures to define the structural orientations. This analysis shows:

- The rock fractures or joints occur either parallel or sub-parallel to mineralised quartz veins; and

- Both, the rock joints and quartz veins, strike principally in the NNW direction and dip towards SW and WSW with dip angles ranging from 47 to 88º. However, the most pre-dominant veins and rock factures dip at angles greater at 80º, while a second recognizable set dips at angles of less than 20º in principally in the same directions as the dominant sets.

In addition, RQD data for the logged cores were provided and SRK estimates that the average RQD is about 70, which indicates a good quality rock mass.

16.2.1 Conclusions

SRK believes for this level of study the amount of data available was sufficient to determine preliminary overall slope angles for the purposes of pit optimisation and conceptual pit design. This involved the development of geotechnical parameters based on the structural data available and SRK's expectation that instability will principally be structurally controlled and therefore, kinematic analysis will be required to determine the optimum bench and inter ramp height and slopes angles in future more detailed studies. It may also be necessary to domain the structures into 'footwall' and 'hangingwall' layouts, as most of localised instability may be planar and critical in the footwall, given that the fact that structures are mostly parallel to sub-parallel to veins or the majority of the exposed pit slope along strike.

The angles determined for the purposes of pit optimisation and conceptual pit design are shown in Table 16-1.

Table 16-1: Open pit slope angles determined by SRK

Overall pit slope angle	Degrees
Footwall	45
Hangingwall	50

16.3 Hydrogeology

SRK reviewed the potential impact of water inflows, as discussed in Section 20.1.

16.4 Seismicity

SRK expects that the that seismic risks are low though additional analysis of seismic data from the regional digital seismic stations is required to determine the design criteria for buildings and pit slopes.

16.5 Open Pit Optimisation for Preliminary Pit Design

SRK used the Whittle 4X pit optimisation software to determine the economic pit limits initially for the Measured and Indicated Resources only and then incorporating the Inferred Resources to understand the upside potential. The key parameters used for the optimisation are summarised in Table 16-2 and in Table 16-2. The metal prices and smelter charges were estimated using recognised sources including the Mining Cost Service (Infomine). The mining cost used in the optimisation is estimated using a base mining cost and an incremental cost for depth. The mining losses and dilution factors were considered suitable for the nature of the geological contacts (clear or gradual), dip and shape of the mineralised zone (continuous or fragmented), mineralised thickness, maximum thickness of interburden and minimum thickness of mineralization, and mining equipment selected. These suggested an average waste dilution factor of 5.0% and ore losses of 3.0%.

The currency used for the purposes of the mining study is United States Dollars (USD). The base case metal prices used in the study are based on consensus market forecasts as follows:

- Gold price of USD 1160 per troy ounce ("oz")
- Copper price of USD 6050 per tonne

Table 16-2: Pit optimisation criteria - general

Item	Unit	Value
Mining losses	%	3.0
Waste dilution	%	5.0
SG mineralization	m^3/t	2.73
SG waste	m^3/t	2.73
Overall pit slope angle (Footwall/Hangingwall)	deg	45/50

Table 16-3: Pit optimisation criteria – processing and economic parameters

Criteria	Unit	Value
Processing - Gold Concentrate to Cyanidation		
Recovery Au	%	42.5
Processing - Gold/Copper Concentrate		
Recovery Au	%	40
Recovery Cu	%	80
Concentrate Grade Cu	%	22.5
Smelter Recovery Au	%	95
Smelter Recovery Cu	%	95
Operating Cost Breakdown		
Reverence Mining Elevation	m	110
Ref Mining Cost Waste	USD/t	3.5
Ref Mining Cost Ore	USD/t_{ore}	3.5
Incremental Mining Cost	USD/t/10m	0.05
Processing Cost Flotation	USD/t_{ore}	12.0
Process Cost Cyanidation	USD/t_{conc}	1.0
G&A Cost	USD/t_{ore}	Included in processing costs
Transport to Hitura Process Plant	USD/tkm	0.28
Transport Distance	km	20
Transportation Cost	USD/t_{ore}	5.60
TC/RC - Copper		
T/C Copper Conc.	USD/t_{conc}	90.0
R/C Copper	USD/lb Cu	0.09
	USD/t Cu	198.42
TC/RC - Gold		
Refining Deduction Au	%	0.5
Refining Charge Au	USD/oz	6.0

The nested pit shells produced by Whittle for the main asset are graphically presented below in Figure 16-1 with the highlighted option indicating the final selected pit shell for conceptual design.

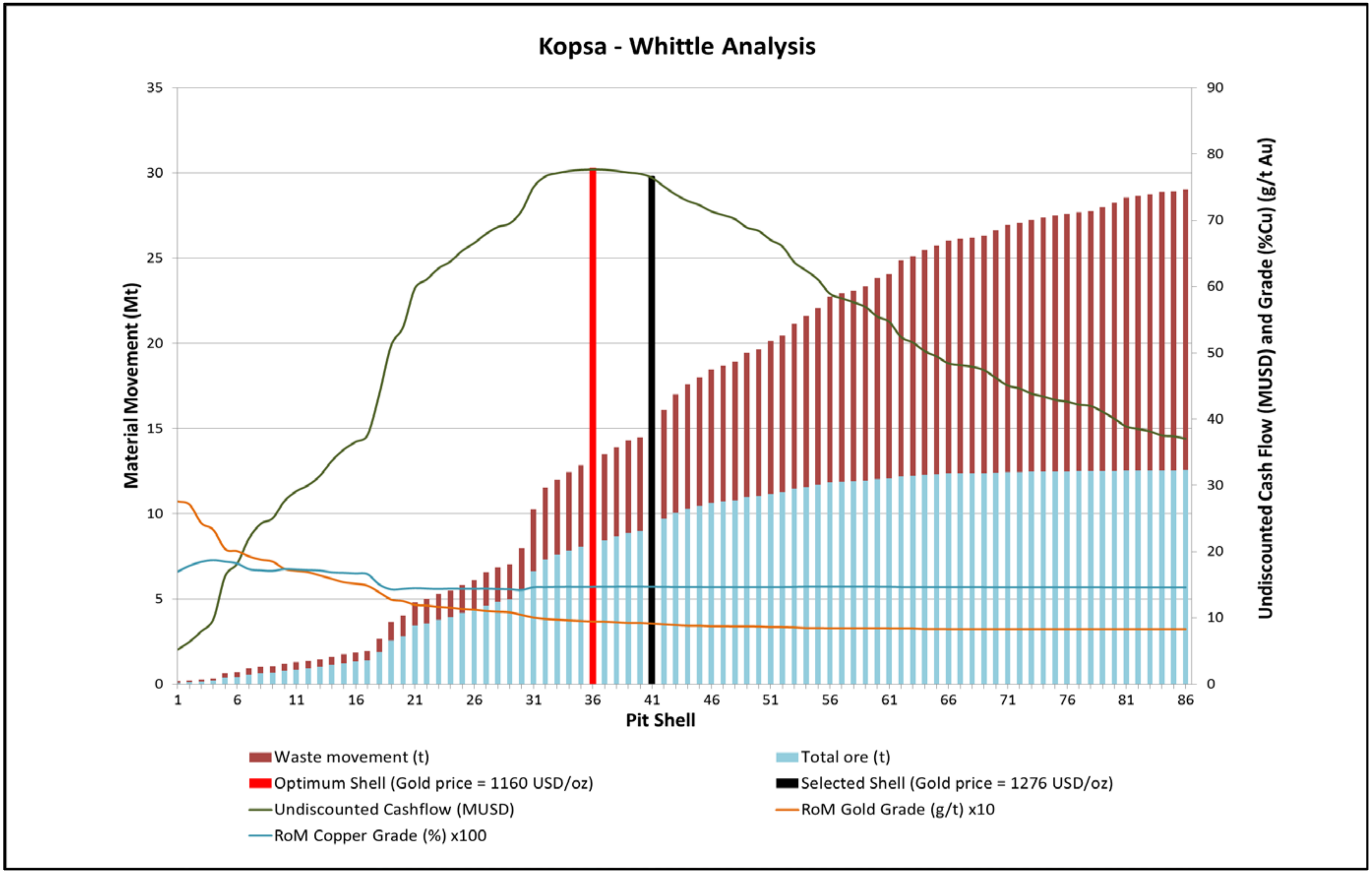


Figure 16-1: Pit optimisation results (Source: SRK, 2013)

While the maximum undiscounted cash flow is achieved by shell 36 with the gold price 1160 USD/oz; SRK selected pit shell 41 with revenue adjustment factor 1.1 and the gold price 1276 USD/oz for the pit design.

16.6 Open Pit design

SRK developed an open pit mine design using a ramp gradient of 10%, which is suitable for the operation of mining trucks. SRK used a standard ramp width of 23 m dropping to 15 m for the final bench.

The pit design assumes the following general design parameters presented in Table 16-4 to achieve the overall slope angles:

Table 16-4: Pit design parameters

Pit design parameters	Units	Value
Footwall (FW)- Overall Slope Angle	deg	45
Hangingwall (HW) - Overall Slope Angle	deg	50
Inter ramp angle	deg	56
Minimum mining width	m	35
Working bench height	m	5
Final bench height	m	20
Final bench slope angle	m	75
Safety berm width	m	8
Decline (ramp) width (one/two way)	m	15/23
Ramp angle	%	10

Figure 16-2 and Figure 16-3 present a plan view and oblique view of the design produced by SRK, whilst Figure 16-4 shows the site layout and the waste rock dump options.

The conceptual Kopsa pit design is approximately 0.7 km long and 0.2 km wide, reaching a maximum depth of 115 m from the surface.

For the waste storage facility ("WSF") design SRK has considered a height for each waste dump stage of 15 m. Future WSF designs may need to consider a greater area than that provided by Belvedere as shown on Figure 16-4 - black dashed line, subject to mass yields at sorting.

SRK notes that the WSF has been conceptually located over an aquifer. The current assumption is that this facility will be lined, which should minimise the risks of leaching to this feature. The location of the WSF should be reviewed subject to the results of future sterilisation drilling, soils geotechnical investigations and confirmation of likely waste rock tonnages.

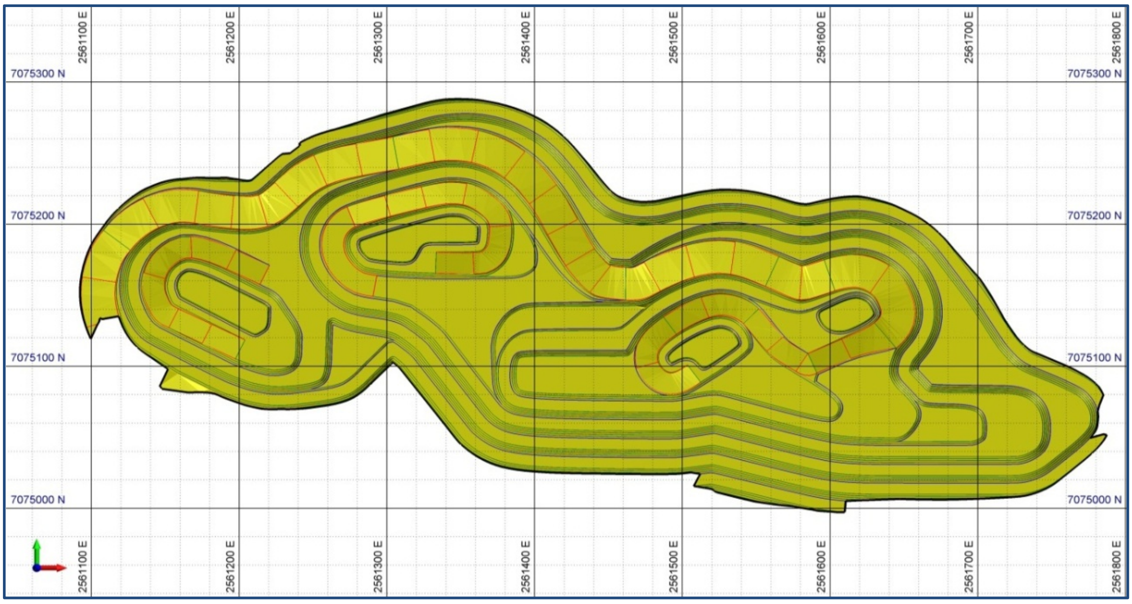


Figure 16-2: Preliminary pit design – Kopsa plan view (Source: SRK, 2013)

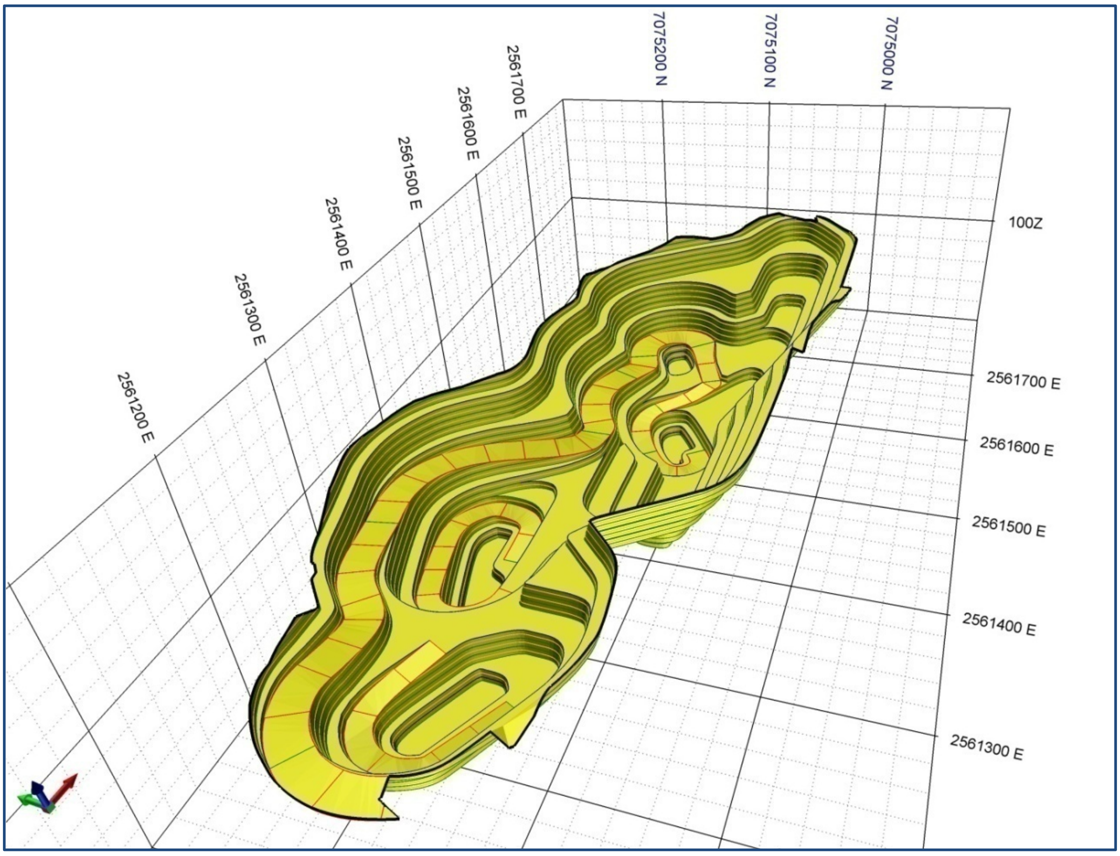


Figure 16-3: Preliminary pit design – Kopsa oblique view (Source: SRK, 2013)

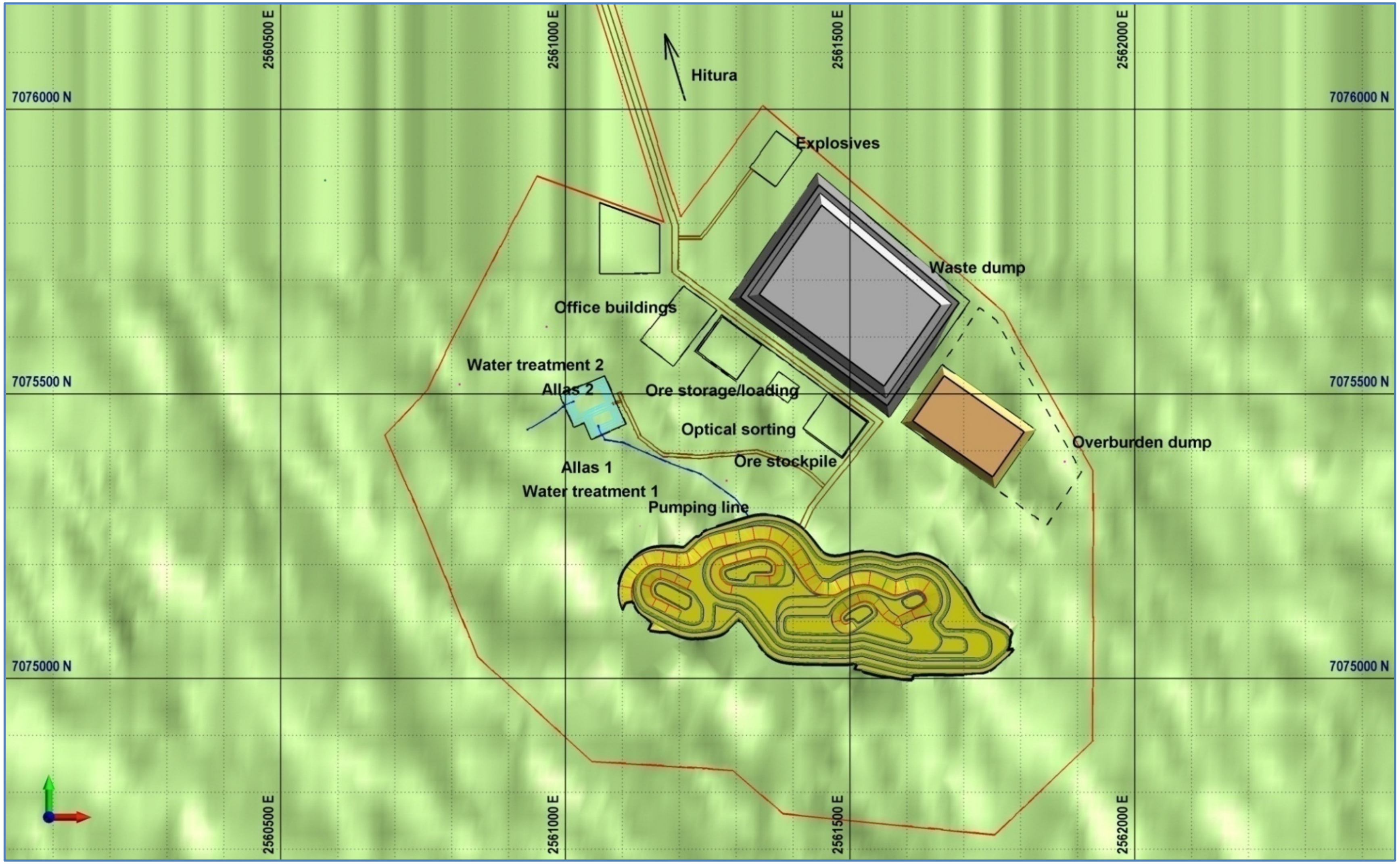


Figure 16-4: Preliminary site layout* – Kopsa plan view (Source: Modified from belvedere 2013)

**Red line – concession boundary, brown lines – planned roads, dashed black line – planned overburden dump area*

16.7 Life of Mine Plan

SRK provided a number of schedules with different mining rates to determine the optimum scenario with and without a sorting processing option. Mine schedules for 0.5, 0.75, 1.0 and 1.2 Mtpa were produced.

The mine plan as presented in Table 16-5 is based on a production rate of 1.2 Mtpa which generates the highest project NPV and best mining scenario, with an overall mine life of 8 years. SRK considered a mining sequence based on three push-backs, each containing some 1.6 to 3.7 Mt of mineralised material or 2 to 5 years life. The basic mining schedule was constrained to a maximum of 6 benches (30 m) per year and there were typically 1 or 2 cut-backs being developed at any one time.

SRK split the mineralised material into three categories using gold equivalent grade ("Au EQ"). The formula to calculate the equivalent gold grade:

$$(\text{Au EQ})\ (\text{g/t}) = 0.982830 \times \text{Au}\ (\text{g/t}) + 1.672011 \times \text{Cu}\ (\%)$$

The three categories are based on cut-off grade calculations as follows:

- High-grade: Au EQ > 0.79 g/t. This is processed as it is mined.
- Low-grade (Marginal): 0.66 g/t < Au EQ < 0.79 g/t. This is stockpiled and processed at the end of LOM.
- Mineralised waste: 0.46 g/t < Au EQ < 0.66 g/t.

Mineralised waste is stockpiled for possible processing later if the gold price increases. It is not processed in production schedule.

SRK also classified the mineralised material (as defined by the SRK mineralisation wireframes - not including S-rich waste) into two categories using sulphide sulphur content within the Finnish legislation (EU Directive 2006/21/EC – Management of Waste from Extractive Industries)

The two categories are:

- Sulphide material: S > 1.0 % classified as non-inert
- Non Sulphide material: S < 1.0 % classified as inert

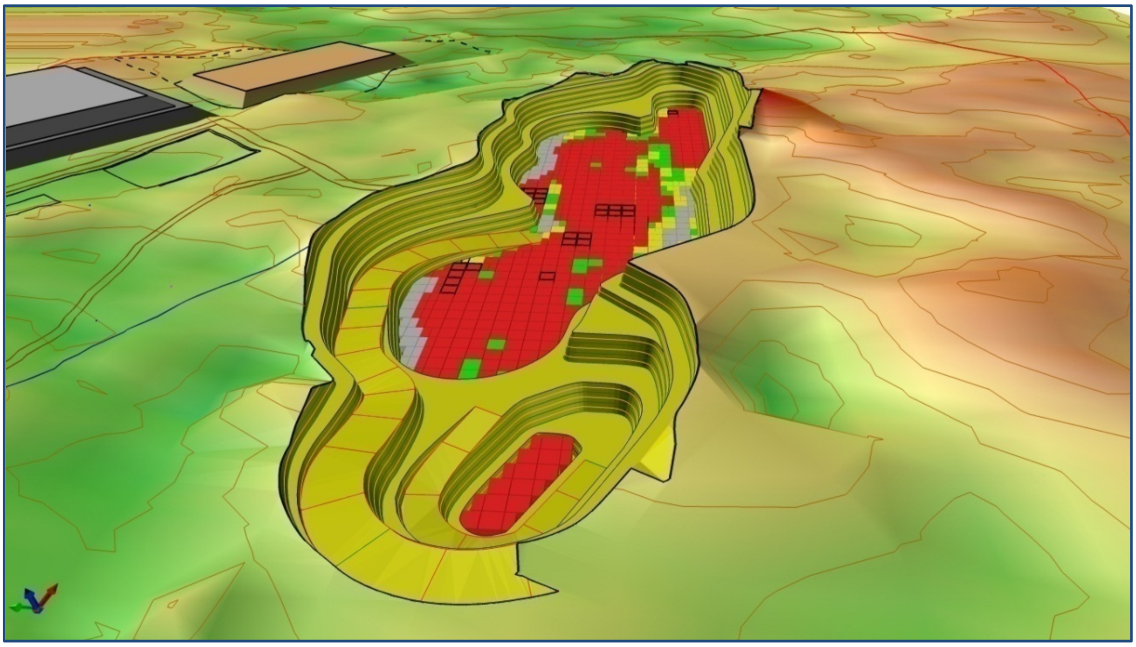


Figure 16-5: Working bench by material classes* – Kopsa oblique view (Source: SRK, 2013)

**Red blocks – High grade material, green blocks – Low grade material, yellow blocks – mineralised waste, black contour blocks – Sulphide material*

SRK scheduled production on a bench by bench basis into the cutbacks. During operation, it will be possible to develop cutbacks to maintain ore-waste ratios across the year. SRK limited total rock mass production in each year based on the number of units within production ramp-up at 70% in the first year and full capacity from then on.

The result for mining schedule is shown in Figure 16-6 and Table 16-5.

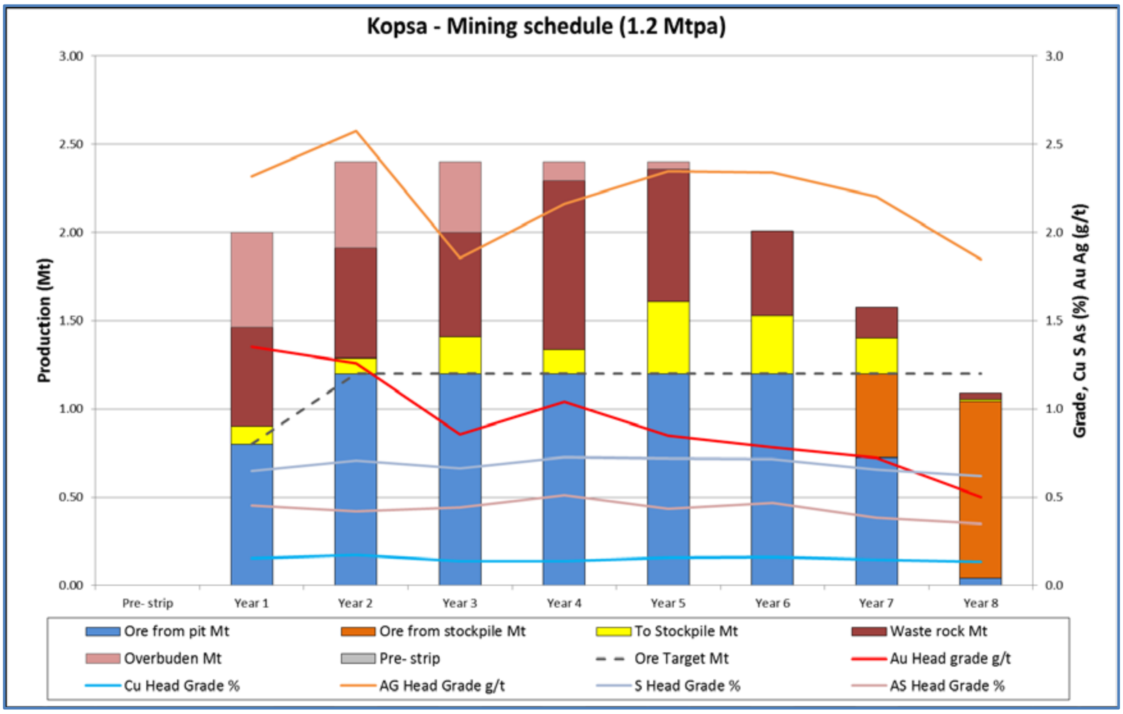


Figure 16-6: Production schedule (Source: SRK, 2013)

Table 16-5: Production schedule – Kopsa (Measured and Indicated Resources only)

		Pre-s	Year 1	Year 2	Year 3	Year 4	Year 5	Year 6	Year 7	Year 8
TOTAL ROCK	**Mt**	**14.80**	**2.00**	**2.40**	**2.40**	**2.40**	**2.40**	**2.01**	**1.10**	**0.09**
TOTAL WASTE	**Mt**	**5.75**	**1.10**	**1.11**	**0.99**	**1.06**	**0.79**	**0.48**	**0.17**	**0.04**
Waste rock	Mt	4.18	0.56	0.62	0.59	0.96	0.75	0.48	0.17	0.04
Overbuden	Mt	1.57	0.54	0.49	0.40	0.10	0.04	-	-	-
To Stockpile	Mt	1.48	0.10	0.09	0.21	0.14	0.41	0.33	0.20	0.01
SP Sulph	Mt	0.008	-	0.001	0.006	0.000	0.001	-	-	-
SP Non Sulph	Mt	1.47	0.10	0.09	0.20	0.13	0.41	0.33	0.20	0.01
Stockpile Au grade	g/t	0.49	0.52	0.53	0.53	0.52	0.46	0.49	0.49	0.45
Stockpile Cu grade	%	0.14	0.12	0.12	0.11	0.12	0.16	0.13	0.14	0.15
Stockpile Ag grade	g/t	1.83	1.57	1.63	1.56	1.85	2.01	1.81	1.98	2.29
Stockpile S grade	%	0.62	0.54	0.59	0.59	0.66	0.64	0.64	0.63	0.54
Stockpile As grade	%	0.35	0.25	0.29	0.32	0.32	0.38	0.39	0.37	0.20
Strip Ratio	t/t	0.64	1.22	0.86	0.70	0.80	0.49	0.31	0.19	0.78
Mined	t	7.57	0.80	1.20	1.20	1.20	1.20	1.20	0.73	0.04
Sulph	t	0.22	0.01	0.05	0.02	0.07	0.06	0.02	0.00	-
Non Sulph	t	7.35	0.79	1.15	1.18	1.13	1.14	1.18	0.72	0.04
Mined Au grade	g/t	0.99	1.35	1.26	0.86	1.04	0.85	0.78	0.88	0.64
Mined Cu grade	%	0.15	0.15	0.17	0.14	0.14	0.16	0.16	0.15	0.15
Mined Ag grade	g/t	2.28	2.32	2.58	1.85	2.16	2.35	2.34	2.46	2.34
Mined S grade	%	0.70	0.65	0.71	0.67	0.73	0.72	0.72	0.68	0.55
Mined As grade	%	0.45	0.45	0.42	0.44	0.51	0.44	0.47	0.40	0.23
Au Mined	t	7.49	1.08	1.51	1.03	1.25	1.02	0.94	0.63	0.03
Cu Mined	t	11.67	1.22	2.07	1.67	1.68	1.91	1.94	1.11	0.06
Ag Mined	t	17.27	1.85	3.09	2.22	2.59	2.82	2.81	1.79	0.09
S Mined	t	52.74	5.21	8.49	7.99	8.71	8.62	8.60	4.91	0.22
As Mined	t	34.01	3.64	5.05	5.31	6.16	5.22	5.61	2.94	0.09
Processed	**Mt**	**9.04**	**0.80**	**1.20**	**1.20**	**1.20**	**1.20**	**1.20**	**1.20**	**1.04**
Material from pit	Mt	7.57	0.80	1.20	1.20	1.20	1.20	1.20	0.73	0.04
Material from stockpile	Mt	1.48	-	-	-	-	-	-	0.47	1.00
Au Head grade	**g/t**	**0.91**	**1.35**	**1.26**	**0.86**	**1.04**	**0.85**	**0.78**	**0.72**	**0.50**
Cu Head Grade	%	0.15	0.15	0.17	0.14	0.14	0.16	0.16	0.15	0.14
AG Head Grade	g/t	2.21	2.32	2.58	1.85	2.16	2.35	2.34	2.20	1.85
S Head Grade	%	0.69	0.65	0.71	0.67	0.73	0.72	0.72	0.66	0.62
AS Head Grade	%	0.43	0.45	0.42	0.44	0.51	0.44	0.47	0.38	0.35
Au Metal	**t**	**8.22**	**1.08**	**1.51**	**1.03**	**1.25**	**1.02**	**0.94**	**0.87**	**0.52**
Cu Metal	t	13.66	1.22	2.07	1.67	1.68	1.91	1.94	1.75	1.41
AG Metal	t	19.97	1.85	3.09	2.22	2.59	2.82	2.81	2.65	1.93
S Product	t	61.95	5.21	8.49	7.99	8.71	8.62	8.60	7.86	6.48
AS Product	t	39.22	3.64	5.05	5.31	6.16	5.22	5.61	4.60	3.64

It can be seen in Figure 16-6 that the mill feed continues after mining has ceased, as the low grade stockpile material is run down prior to mine closure.

The low grade material is stockpiled separately with maximum stockpiled tonnage 1.3 Mt, enabling higher grade material to be recovered first.

Mineralised material distribution by cut-backs and gold equivalent grade is presented in Figure 16-7.

SRK notes that this schedule may be constrained if blending is needed to balance sulphur, as well as to minimise arsenic content in the concentrate.

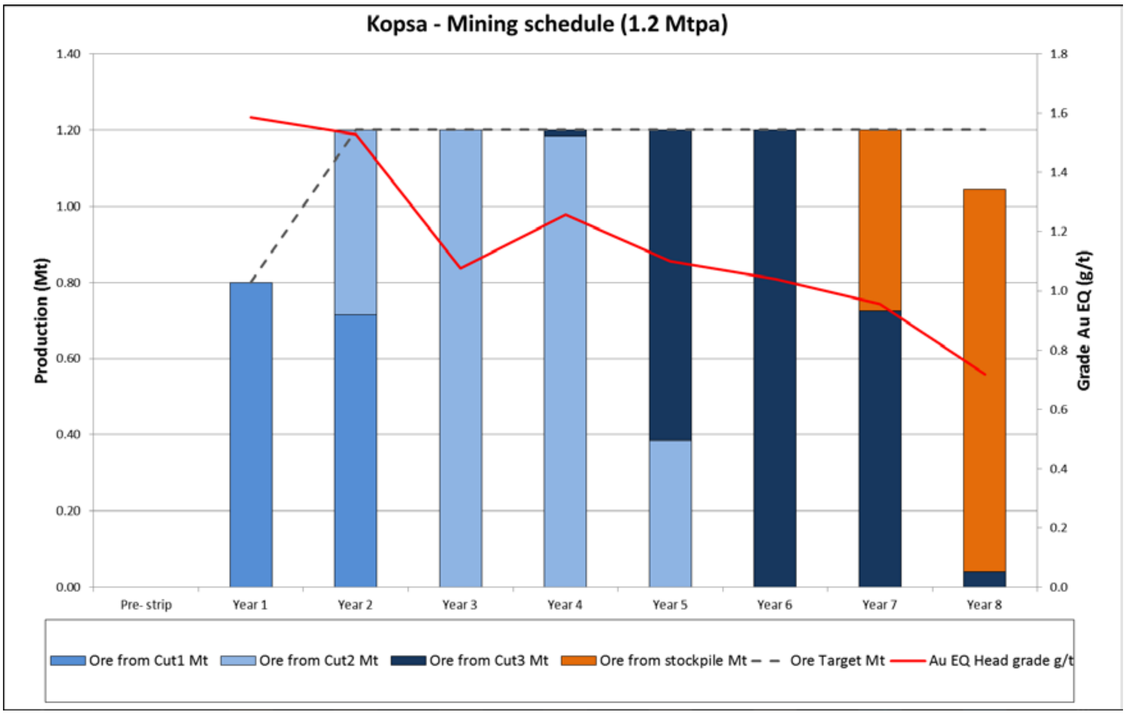


Figure 16-7: Production by cut-backs (Source: SRK, 2013)

16.8 Operating Strategy

The operating strategy is based on the mine schedule to provide:

- a preliminary estimate of mining equipment requirements;
- a preliminary estimate of mining personnel; and
- a basis of the mine cost estimate.

Equipment requirements have been determined using the following assumptions and methods:

- 261 workings days per year and 16 working hours per day (Table 16-8);
- truck and excavator requirements calculated based on productivities and cycle times;
- 3 m^3 capacity excavators and 24 t articulated trucks have been assumed for rock mass movement

- drilling requirements have been based on 5 m benches with 115 mm diameter blasthole drills for the mineralised zones and 10 m benches with 152 mm diameter blasthole drills for the waste;
- ancillary equipment has been based on material movement and primary fleet requirements;
- it has been assumed that the mineralised material from Kopsa pit will transported to the processing facility by the use of a 6 m^3 wheel loader and 40 t on-road trucks

The mine equipment requirements and productivity measured in tonnes per hour and the impact on truck requirements are shown in Figure 16-8.

The mobile and auxiliary equipment requirements are shown on an annual basis in Table 16-6.

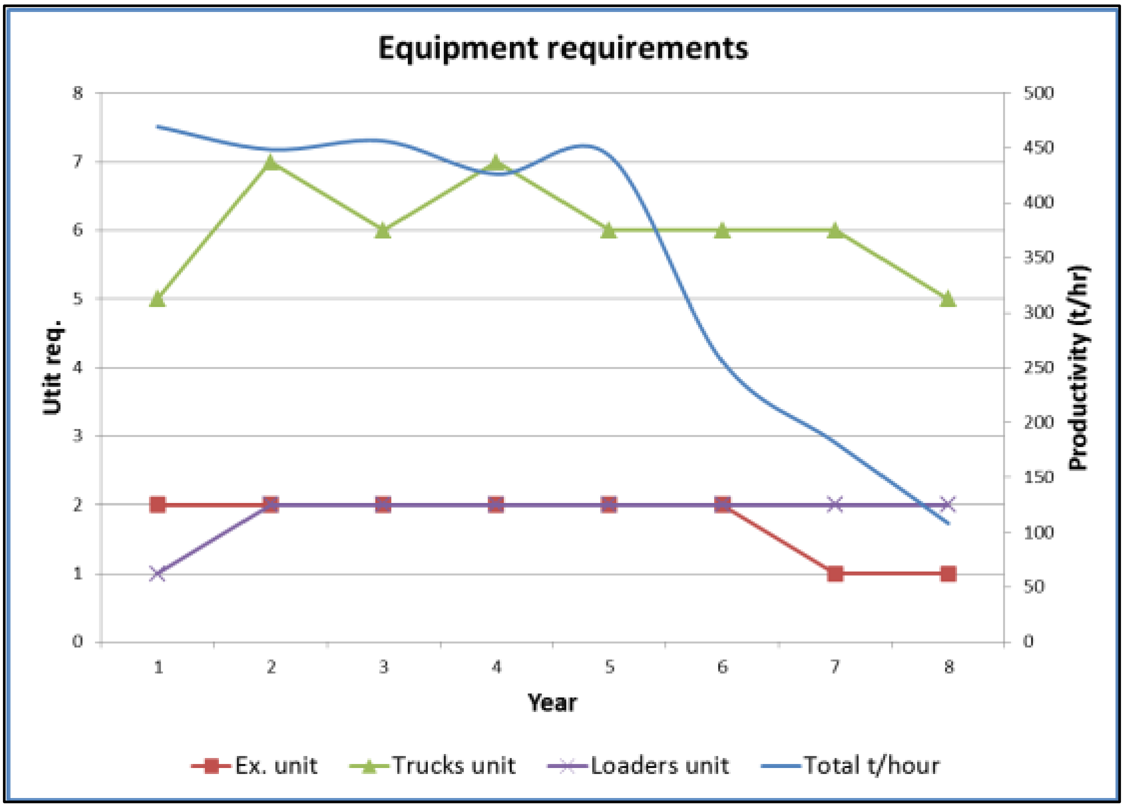


Figure 16-8: Equipment requirements (Source: SRK, 2013)

Table 16-6: Equipment requirements for base case (Scenario 6), year and number of units

Year	1	2	3	4	5	6	7	8
Mobile Mining Equipment								
Ore Percussion drill rig	1	1	1	1	1	1	1	1
Waste rotary rig	1	1	1	1	1	1	1	1
Hydraulic Shovel 3m^3	2	2	2	2	2	2	1	1
24t truck	5	7	6	7	6	6	6	5
40t truck	3	3	3	3	3	3	3	3
Cat D8 type Bulldozer	3	3	3	3	3	3	2	2
GRADER	1	1	1	1	1	1	1	1
Wheeled Loader 6m^3	1	2	2	2	2	2	2	2
W/Bowser	1	1	1	1	1	1	1	1
Auxiliary Equipment								
Tractor & trailer	1	1	1	1	1	1	1	1
Explosives Truck	1	1	1	1	1	1	1	1
Light Tower & gen set	4	4	4	4	4	4	4	4
Hydraulic rock breaker	2	2	2	2	2	2	2	2
Diesel pump 150mm	3	3	3	3	3	3	3	3
Pick up twin cab	2	2	2	2	2	2	2	2
Pick up single cab	4	4	4	4	4	4	4	4
Fuel & Lube Truck	1	1	1	1	1	1	1	1
Low bed and tractor	1	1	1	1	1	1	1	1
Service truck with Hi-ab	1	1	1	1	1	1	1	1
180 psi compressor	2	2	2	2	2	2	2	2
Rough terrain Hi-ab truck	1	1	1	1	1	1	1	1
3t tyre handler	1	1	1	1	1	1	1	1
Crew bus	1	1	1	1	1	1	1	1
Fuel Bowser	1	1	1	1	1	1	1	1
Road wagon	1	1	1	1	1	1	1	1

Personnel requirements have been based on:

- material movements; and
- equipment requirements;

An estimate of the mine staff and maximum personnel required for the life of mine is shown in Table 16-7.

Table 16-7: Personnel required

Mine staff	Number
Mine Manager	1
Maint. Supt	1
Shift Foreman	2
Mine Trainer	1
Workshop Supervisor	2
Senior Planning Engineer	1
Planning Engineer	1
Senior geologist	1
Shift geologist	2
Senior Surveyor	1
Survey Asst.	2
Welders	2
Fuel & Lube	2
Tyre	2
Maint. Planner	1
Service Crew	2
Blasting Gang	2
Drillers	4
Shovel Operators	4
Truck Drivers -25t	14
Truck Drivers -40t	14
Dozer Operators	6
Grader Operators	2
Wheel Loader Operators	4
Water Truck Operators	2
Fitters	15

16.9 Equipment

SRK estimated mining capital costs using the following approach:

- Truck cycle times for ore, waste and overburden are based on the average location of the benches in the pit for each cut-back.
- Based on typical productivities for a 24 t articulated truck with matching excavator and drills, and an average operating time of 2,731 hours per year as shown in Table 16-8.
- Initial capital expenditure is defined as the investment in the first two years to achieve full production No replacement has been planned due to the relatively short life of the operation, the fact that material movement declines in the later years and no sustaining capital expenditure is required. Mining equipment capital costs are presented in Table 16-9. Total capital costs related to mining are shown in Table 16-10.

Table 16-8: Average Operating Time – Kopsa

Item		Units	Value
Generalised Shift Times			
Calendar Days		(days)	365
Days per week		(days)	5
Available Days		(days)	261
Holidays			
Weather			10
Scheduled days		(days)	251
Shifts/day		shifts	2
Annual Work Shifts		shifts	502
Hours/day		hrs/day	16
Scheduled Hrs			4 016
Shift Breakdown			
Overall Shift Pattern		(hrs)	8
Shift Change		Min/Shift	30
Lunch/Coffee Break		Min/Shift	30
Fuelling		Min/Shift	15
Blasting		Min/Day	30
Maximum Work Hours per Day		(hrs)	13.0
		hrs/shift	6.5
Mechanical Efficiency		%	85%
Mining Utilisation		%	79%
Incl.	Shift stoppages	%	81%
Incl.	Effective work	mins/hr	58 (97%)
Work hours per shift		hrs/shift	4.3
Total		hrs/yr	8 760
Available		**hrs/yr**	**4 016**
Mechanical Efficiency (85%)		Hrs	**3 414**
Utilisation (80%)		Hrs	**2 731**

Table 16-9: Mining Equipment Capital Cost – Kopsa

Main equipment	Unit cost (USD)	Costs (USD)
Ore Percussion drill rig	653 072	653 072
Waste rotary rig	437 621	437 621
Hydraulic Shovel 3m3	604 095	1 208 190
24t truck	339 128	2 373 896
40t truck	206 000	618 000
Cat D8 type Bulldozer	624 431	1 873 293
CAT 12M GRADER	277 328	277 328
Wheeled Loader 6m3	612 902	1 225 804
CAT W/Bowser	307 455	307 455
Sub Total		**8 974 659**
	Auxiliary Equipment	
Tractor & trailer	61 800	61 800
Explosives Truck	606 258	606 258
Light Tower & gen set	22 706	90 824
Hydraulic rock breaker	98 159	196 318
Diesel pump	14 626	43 878
Pick up twin cab	61 800	123 600
Pick up single cab	41 200	164 800
Fuel & Lube Truck	83 173	83 173
Low bed and tractor	144 200	72 100
Service truck with Hi-ab	164 285	164 285
Compressor	25 750	51 500
Rough terrain Hi-ab truck	82 400	82 400
3t tyre handler	67 980	67 980
Crew bus	92 700	92 700
Fuel Bowser	83 173	83 173
Road wagon	92 700	92 700
Sub Total		**2 077 489**
Total		**11 052 148**

Table 16-10: Total mining capital costs

Item	Unit	Value
Mine Facilities & Haulage Dispatch System	**(USD)**	6 124 049
Haul Roads	**(USD)**	742 933
Mobile Mining Equipment	**(USD)**	8 974 659
Auxiliary Equipment	**(USD)**	2 077 489
Total	**(USD)**	**17 919 131**

16.10 Labour

SRK use benchmarked annual salaries for mining personnel (Table 16-11). Personnel requirements from Section 16.8 (Table 16-7) have been used to determine the associated operating costs.

Table 16-11: Labour costs – Kopsa

Mine Staff	(EUR/year)	USD/year
Mine Manager	148 000	197 333
Maint Supt.	99 000	132 000
Shift Foreman	44 000	58 667
Mine Trainer	49 000	65 333
Workshop Supervisor	67 000	89 333
Senior Planning Engineer	91 000	121 333
Planning Engineer	66 000	88 000
Senior geologist	91 000	121 333
Shift geologist	66 000	88 000
Senior Surveyor	91 000	121 333
Survey Asst.	38 000	50 667
Welders	46 000	61 333
Fuel & Lube	46 000	61 333
Tyre	46 000	61 333
Maint Planner	49 000	65 333
Service Crew	49 000	65 333
Blasting Gang	43 000	57 333

Statutory social costs required in Finland have been included in the employee salaries are shown in Table 16-12, as well as shift allowances of USD 5,788 per month and vacation salary which is 5% of the annual salary.

Table 16-12: Statutory social costs required in Finland

Operators & Fitters	(EUR/year)	USD/year
Drillers	53 000	70 667
Shovel Operators	53 000	70 667
Truck Drivers -25t	53 000	70 667
Truck Drivers -40t	53 000	70 667
Dozer Operators	53 000	70 667
Grader Operators	53 000	70 667
Wheel Loader Operators	53 000	70 667
Water Truck Operators	53 000	70 667
Fitters - Shifts	46 000	61 333
Fitter Assistants	41 000	54 667

16.11 Unit operational costs

The total operating costs by category over LoM, and average unit operating costs per tonne rock mass moved are shown in Table 16-13. Hauling material to the Hitura plant and labour are the largest contributors to the operating costs.

Table 16-13: Operational costs – Kopsa

Item	Total Cost LoM (M USD)	Average Cost per Tonne (USD/t)
Drilling	0.6	0.04
Blasting	3.5	0.24
Loading	3.7	0.25
Hauling In pit	5.1	0.34
Stockpile Excavation	2.0	0.14
Hauling mine to plant	7.2	0.49
Mobile Mining Equipment	5.7	0.39
Auxiliary Equipment	2.9	0.20
Labour	51.7	3.49
Mine Facilities & Other	4.5	0.30
Total	**86.9**	**5.88**

16.12 Recommendations

SRK recommends that a detailed topographic survey be carried out over the Project area and combined with current and future drill data to determine overburden volume to clarify operational costs for waste movement. In addition, the Project will require further development of the mine block model, pit shells, mine production plan, operations and infrastructure requirements.

17 RECOVERY METHODS

17.1 Process Plant

Belvedere intends to process the Kopsa material through its Hitura flotation mill, located approximately 19 km from the Kopsa site. The Hitura mill until recently processed nickel sulphide ore, at a nominal annual throughput rate of 600 ktpa.

A schematic flowsheet of the Hitura plant is shown in Figure 17-1. The circuit consists of a two stage crushing circuit feeding a three stage milling circuit (rod mill, ball mill, ball mill) ahead of flotation. When treating nickel sulphide ore, the flotation circuit has been configured to produce either one or two concentrate products.

In order to treat Kopsa ore, the flotation circuit would be configured to produce two sulphide concentrates, a marketable copper concentrate, containing some gold and silver, and the bulk sulphide concentrate for further processing on site. The aim of the flowsheet would be to produce a flotation tailings essentially devoid of arsenic, such that it can be stored in the existing Hitura TMF.

The bulk sulphide concentrate would be cyanide leached for the recovery of gold and silver. As indicated by the testwork, the concentrate would be reground ahead of cyanidation. Cyanidation would be followed by a conventional Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL) format, producing a smelted gold/silver doré. The tailings from cyanidation would be subjected to cyanide detoxification ahead of storage in a dedicated facility.

Ore sorting is being considered as part of a mine site facility that conceptually would reduce the amount of ore to be trucked between the Kopsa mine site and Hitura plant site.

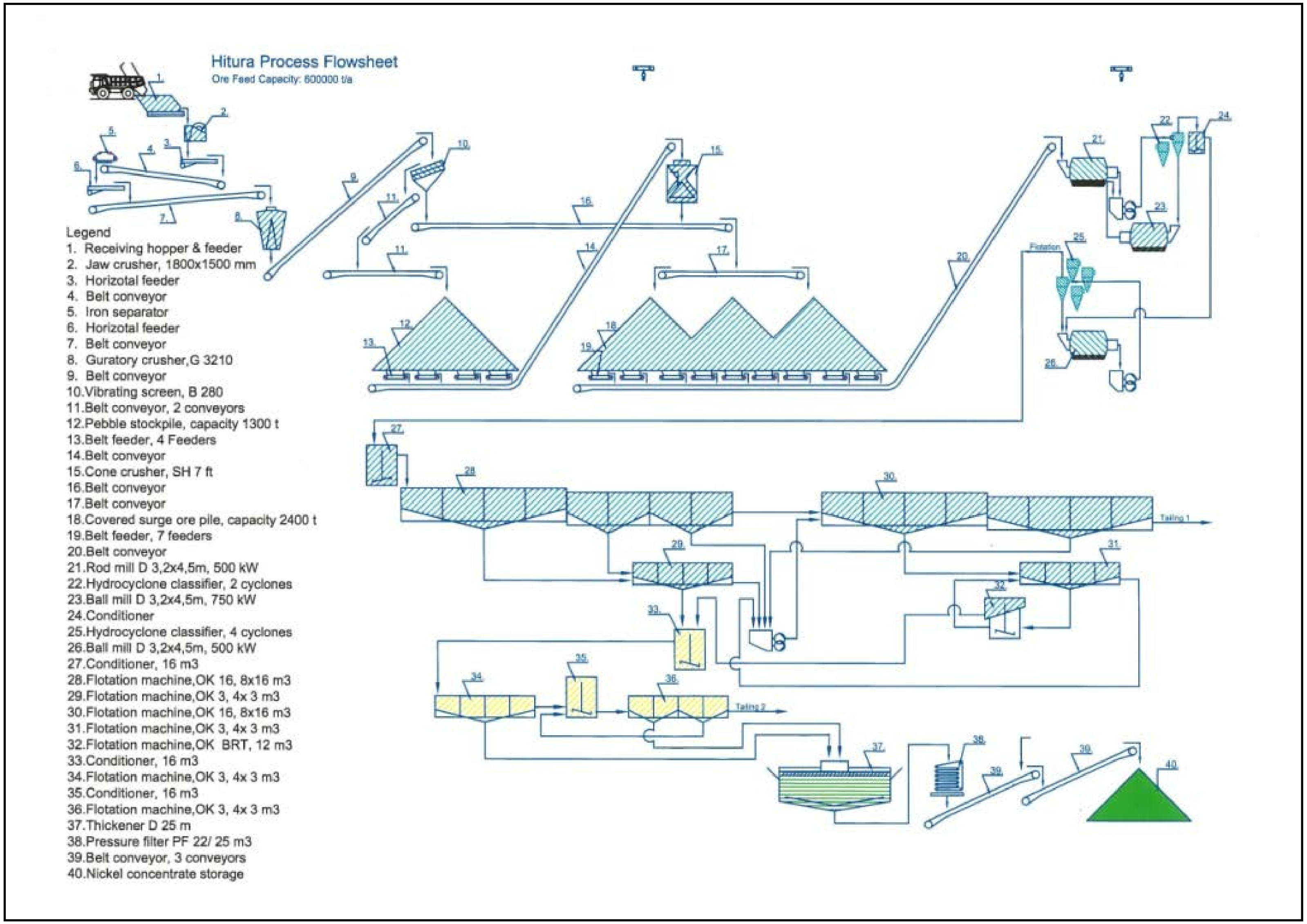


Figure 17-1: Hitura Plant Schematic Flowsheet

17.2 Process Design Criteria

Based on the testwork results received to date, SRK has developed the following design criteria for the processing of the Kopsa material. Two options are presented in Table 17-1, the first for as-mined material (after crushing) delivery to Hitura, and the second with the incorporation of sorting at the mine site.

Table 17-1: Process Design Criteria

Item		Unit	Without Sorting	With Sorting
RoM Production		tpa	500,000	1,200,000
Material delivery to Plant		tpa	500,000	420,000
Sorting Loss	Cu	%	-	25
	Au	%	-	10
Flotation Feed Grade	Cu	%	0.15	0.32
	Au	g/t	0.91	2.34
Copper Concentrate		tpa	2,700	4,800
	Cu Rec	%	80	80
	Au Rec	%	40	40
	Cu	%	22.5	22.5
	Au	g/t	75	82
Sulphide Concentrate		tpa	15,000	12,600
	Au Rec	%	44.75	44.75
	Au	g/t	13.5	35.0
Cyanidation Recovery	Au	%	95	95
Recovery to Doré	Au	%	42.5	42.5
Overall Recovery	Cu	%	80	60
	Au	%	84.75	76.30

The design criteria is based on a number of assumptions, detailed as follows:

- The throughput values are as specified by Belvedere. While the nominal capacity of the Hitura plant is reported as being of the order of 600 ktpa, the lower throughputs shown in Table 17-1 are a reflection both of the size of the Kopsa deposit, and of the perceived harder nature of the Kopsa material. An analysis of the available grinding power in the Hitura plant suggests that, particularly at the higher BWi figures reported for the Kopsa material (20 and 23 kWh/t), the Hitura grinding circuit may not be able to process 500 ktpa of ore, assuming that all of the ore has to be ground to the final grind size (80% -45 μm, as per the recent SGS testwork). These throughput figures will be reviewed as the project progresses, with the collection of further BWi data, and as the flotation flowsheet is optimised, potentially with the inclusion of staged grinding, such that only a small fraction of the incoming plant feed (i.e. a rougher concentrate) needs to be ground to the final grind size.

- The response of the Kopsa material to sorting is based on the testwork conducted to date, with the following specific assumptions:
  - The 65% mass rejection is based on the sorting testwork results without considering the +40 mm fraction, i.e. it is based on the XRT results for the -40 mm fractions plus the unsorted -8 mm fraction;
  - The Cu recovery (75%, i.e. 25% loss) is based on the results for the entire sample (67%, as described in Section 13.2.3), increased slightly to account for the higher Cu grade of the Resource over the sample tested; and
  - The Au recovery (90%) is based on the same set of results, with no discount applied despite the higher grade of the sample tested over the Resource grade.

These values could therefore be considered as somewhat optimistic. Further testwork will be undertaken as the project progresses to optimise the sorting stage, particularly focussing on the use of material with Cu and Au grades closer to the Mineral Resource average, and with crushing of the material to a top size appropriate to the sorting method to be used (i.e. 32-40 mm for XRT sorting).

- The flotation criteria assume fixed Cu and Au recoveries to a fixed grade of Cu concentrate, and fixed mass and Au recoveries to the sulphide concentrate. These figures are assumed to be the same for both the non-sorting and sorting cases, i.e. they do not vary with the different plant feed grades for each scenario. The figures used are based on the SGS testwork, although this testwork has yet to definitively demonstrate the simultaneous production of a high Cu, low As (i.e. below penalty limits) Cu concentrate and a low As final tailing following the production of the sulphide concentrate, and as such the recoveries and mass yield to the sulphide concentrate should be considered as provision al figures at this stage. Again, further flotation optimisation testwork will be undertaken as the project develops, with the specific aim of determining these values to a great degree of precision.
- While the cyanidation testwork conducted by SGS was conducted under intensive cyanidation conditions, SRK believes that the more appropriate flowsheet for the cyanidation stage will be a conventional CIP/CIL circuit, there being in SRK's opinion no compelling process or cost advantage to the use of an intensive cyanidation circuit. Based on SRK's experience with such circuits, SRK believes that a conventional CIP/CIL circuit will be capable of a recovery similar to that achieved in the intensive cyanidation testwork.

17.3 Capital Costs

SRK has estimated capital costs for the conversion of the Hitura process plant to treat Kopsa material as follows:

- Hitura plant reconfiguring and general refurbishment (nominal figure): EUR 500,000
- Sulphide Concentrate CIL plant and associated goldroom: EUR 3,750,000
- Sorting plant (assumes contract crushing): EUR 1,500,000
- Hitura plant expansion (to 1 - 1.2 Mtpa): EUR 7,500,000, i.e. without sorting

The figures for the Hitura plant are estimates based on SRK's internal database. The sorting figure is based on a figure provided by Belvedere per sorting unit (per 500,000 tpa of production capacity).

17.4 Operating Costs

SRK has estimated operating costs for processing of the Kopsa material through the modified Hitura plant, and with the inclusion of sorting, as shown in Table 17-2.

Table 17-2: Process Plant Operating Costs

Item	Cost (EUR/t RoM)
Crushing (non sorting case)	1.00
Sorting (including crushing)	1.75
Grinding	1.90
Flotation	3.10
Cyanidation	0.75
Filtration	0.50
Processing General	0.32
Maintenance	0.32

The costs excluding Sorting and Cyanidation, are based on the 2012 actual costs for the Hitura plant. The sorting cost is based on a figure provided by Belvedere, and the Cyanidation cost is an SRK estimate.

17.5 Recommendations

- Design engineering activities are required to support the next phase of the project's development and should include engineering of the new sections of plant required as additions to the Hitura facility, principally the CIL plant and associated gold recovery processes (elution, goldroom). In addition, a detailed analysis of the existing Hitura facility will be required, in order to estimate the process and engineering modifications required in order to covert the plant from its existing configuration to the configuration required for the Kopsa project. Consideration will also be required as to whether parts of the plant require refurbishment in order to meet contemporary operating requirements.
- The cost for such an engineering programme is likely to be of the order of EUR 1.0-1.5 million.

18 PROJECT INFRASTRUCTURE

18.1 Hitura Plant Site

Being an existing plant site and given that the proposed throughput for the Kopsa operation is of the same order as the historic production rate for the Hitura plant (assuming base case, Scenario 6), the infrastructure requirements for the Hitura plant site will be similar to those required when the plant was previously operating on Hitura nickel ore. Notably therefore, the existing power supply should be adequate for the Kopsa operation. And so this is not commented upon further here. Further, water supply and wastewater treatment requirements to support the Kopsa operation are described in Section 20 of this report and tailings disposal requirements are described in Section 20 of this report.

18.2 Kopsa Mine Site

The area around the Kopsa mine site is well serviced in terms of infrastructure such as water and power, in support of the local communities and farms. As the requirements of the mine site itself will be relatively minor, the mine operations infrastructure requirements should be able to be met from the existing infrastructure in the area.

The sorting option will require additional infrastructure, particularly power, at the mine site. While there should be sufficient power transmission capacity in the vicinity of the mine to support the crushing and sorting operation, a suitable fallback position would be to generate power on site, either using stand-alone generators, or through the use of “self-contained” process units, i.e. units that have their own power source.

18.3 Transportation to Process Plant

The most significant infrastructure requirement for the Kopsa operation will be that needed to support the proposed haulage of material from the Kopsa mine to the Hitura plant, a distance of approximately 20 km. The material will be hauled using 40 t road haulage trucks (see Figure 18-2 for an example), and at peak production (years 2-6), SRK estimates that 7 hourly trips (or 3.5 return trips) will be required, assuming base case (Scenario 6, sorting and ROM production at 1.2 Mtpa), which at this stage the Company anticipates would be the maximum permissible by the authorities, given the permanent dwellings along the proposed transport routes, see Figure 18-1.

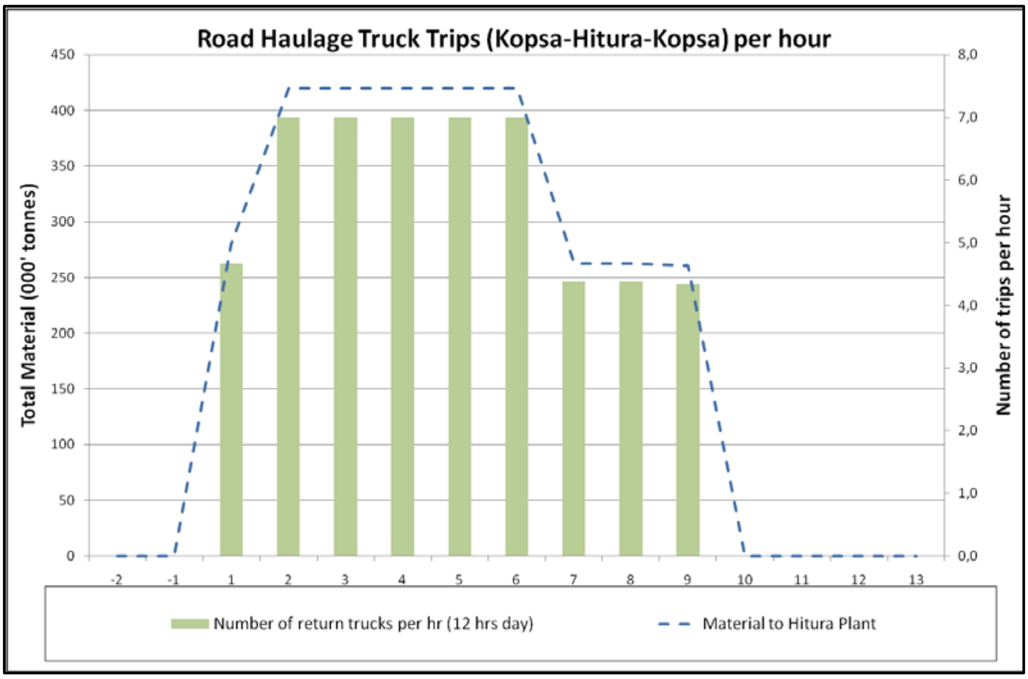


Figure 18-1: Road haulage, estimated hourly one-way trips over the LOM (Source: SRK, 2013)

Figure 18-2: 40 Tonne Road Haulage Truck

Figure 18-3 shows the location of the Kopsa mine site and the Hitura plant site, together with the road haulage route options between the sites. Just to the east of the mine site is the sealed road Tiitonrannantie (route no 7630), and there are three existing unsealed roads that lead off this route into the mine site vicinity. Either of these existing roads could be upgraded to become the mine access road, or a dedicated road could be constructed. Route 7630 continues past the Hitura plant site.

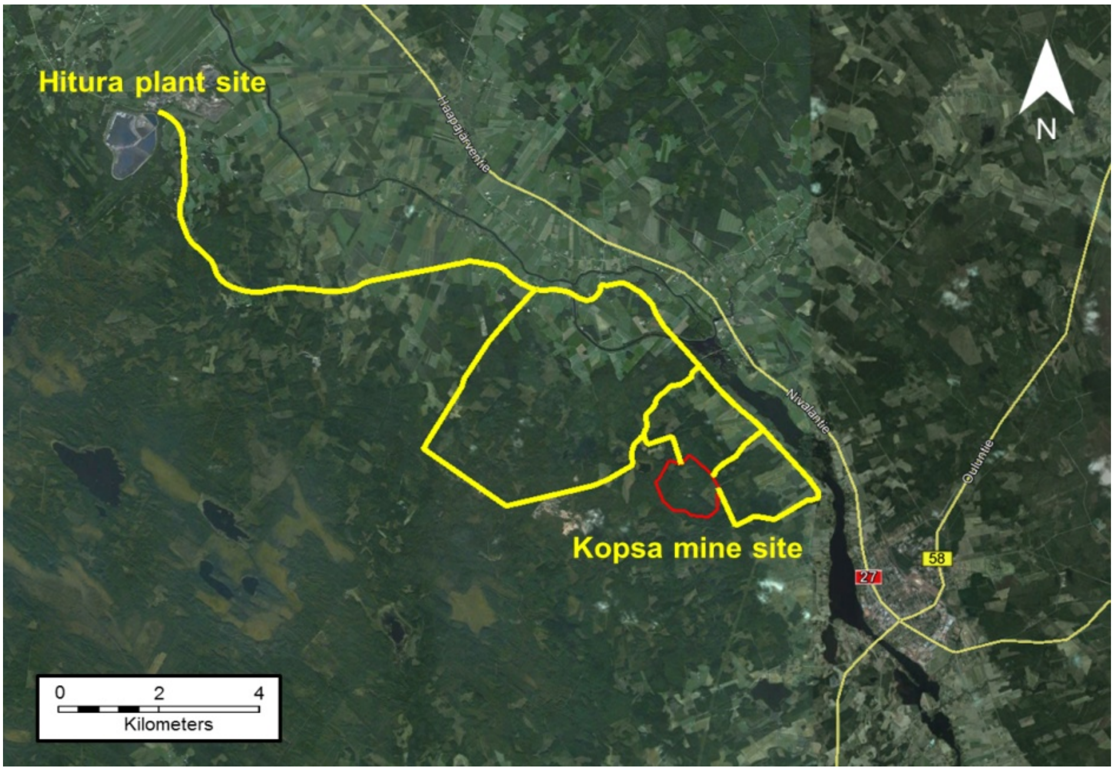


Figure 18-3: Location of Kopsa Mine site, Hitura Plant and Road Haulage Route Options (Source: SRK, 2013)

This route includes one significant river crossing – over the Lassikoski diversion channel – and an analysis would be required to determine the suitability of this bridge for the proposed traffic flow (see Figure 18-4).

Figure 18-4: Lassikoski River Crossing on Route 7630 (Source: SRK, 2013)

An alternative route would be to use the existing unsealed road that leads west from north of the mine site area, which joins the unsealed road Kalkkuperäntie and which then joins Route 7630 at the village of Arvola. While this route is slightly longer than the eastern route, it uses a less trafficked road for a longer portion of the route, and so may result in less of an impact on the local common-use infrastructure and communities. This route also does not involve any significant waterway crossings.

Given the relatively high volume of traffic that the project will introduce to the transport route, significant ongoing stakeholder engagement will be required regarding access to this infrastructure option as the project progresses.

19 MARKET STUDIES AND CONTRACTS

No market studies have been conducted and no contracts have been signed to date.

20 ENVIRONMENTAL STUDIES, PERMITTING AND SOCIAL OR COMMUNITY IMPACT

20.1 Hydrology and Hydrogeology

The hydrological and hydrogeological scope of work carried out for this project includes the following key elements:

- A desktop study including the compilation, review and analysis of data from the Kopsa and Hitura sites;
- Development of preliminary conceptual surface and groundwater model for the mine site;
- Review of the existing model to identify uncertainties and gaps in the present knowledge of the site (gap analysis);
- Development of simple conceptual designs for the mine water management elements of the project;
- Estimation of CAPEX, OPEX and post closure costs for the mine water management elements of the project; and
- A list of recommended activities to advance the project to feasibility study.

This hydrological and hydrogeological study is based on the following sources:

- NI 43-101 Technical Report, Kopsa Gold-Copper Deposit, Central Ostrobothnia, Finland (Belvedere, August 2012)
- NI 43-101 Technical Report, Updated Reserve and Resource Estimate of the Hitura Nickel Mine in Central Finland (Belvedere, December 2012)
- Three-dimensional geological modelling and groundwater flow modelling of the Töllinperä aquifer in the Hitura nickel mine area, Finland – providing the framework for restoration and protection of the aquifer (Artimo et al, 2004)
- Hitura Mine Closure Plan. (Finn Nickel Oy, May 2008)
- Hitura Environment, Health and Safety Annual Report 2012. (Belvedere, July 2013)
- Hitura Ground Survey for New Tailings Reservoirs (Geobotnia, March 2013)
- Baseline Monitoring report for the Kopsa Area (LVT, July 2008)
- Variations in national groundwater level and water quality between 1975-1999 (STYKE, 2001)
- Belvedere-Resources.com (Belvedere, July 2013)
- Exploration Drillhole Data, Groundwater Quality 2011 - 2012 (Nablabs laboratories, November.2012)
- Lepola Groundwater Quality Data 1976-2012 (Belvedere, 2013).

20.1.1 Hitura Tailings Storage Facility and Surrounding Area

According to previous ground surveys conducted at the Hitura site by Geobotnia Ltd and others, most of the ground that underlies and surrounds the Hitura Tailings Storage Facility (TMF) comprises moraine containing a mixture of peat, sandy silts and glacial till (clay and boulders). There are no records of hydraulic tests having been performed on this moraine, but nevertheless the suggestion has been that the material in the moraine has a low hydraulic conductivity and this assumption would certainly be consistent with the descriptions of the materials given in the reports. Notwithstanding this, there are also two significant groundwater features at the site; namely the north-west orientated Weichselian esker system that passes under the western half of the existing facility and the Töllinperä Aquifer which is located to the south of the facility. This aquifer was a source of potable water for the local community (Figure 20-1).

Records reveal that approximately 30 years ago there was an uncontrolled release of leachate from the TMF into the underlying esker. This impacted the northern part of the TMF and remediation work was undertaken to reduce the elevated levels of sulphate, nickel and chloride in the groundwater. Elevated sulphate concentrations were also later discovered in the southeastern area and confirmed contamination of parts of the Töllinperä Aquifer. As a result, the TMF and surrounding area was subjected to extensive geophysical studies and numerical groundwater modelling in an attempt to create an effective remediation strategy. Seepage from the TMF is now captured by a combination of groundwater wells and a system of deep seepage ditches located around the periphery of the facility and is re-circulated back into the pond. More recent water quality monitoring of the site suggests that this remedial solution has successfully isolated the TMF from the underlying aquifer (Artimo et al., 2004).

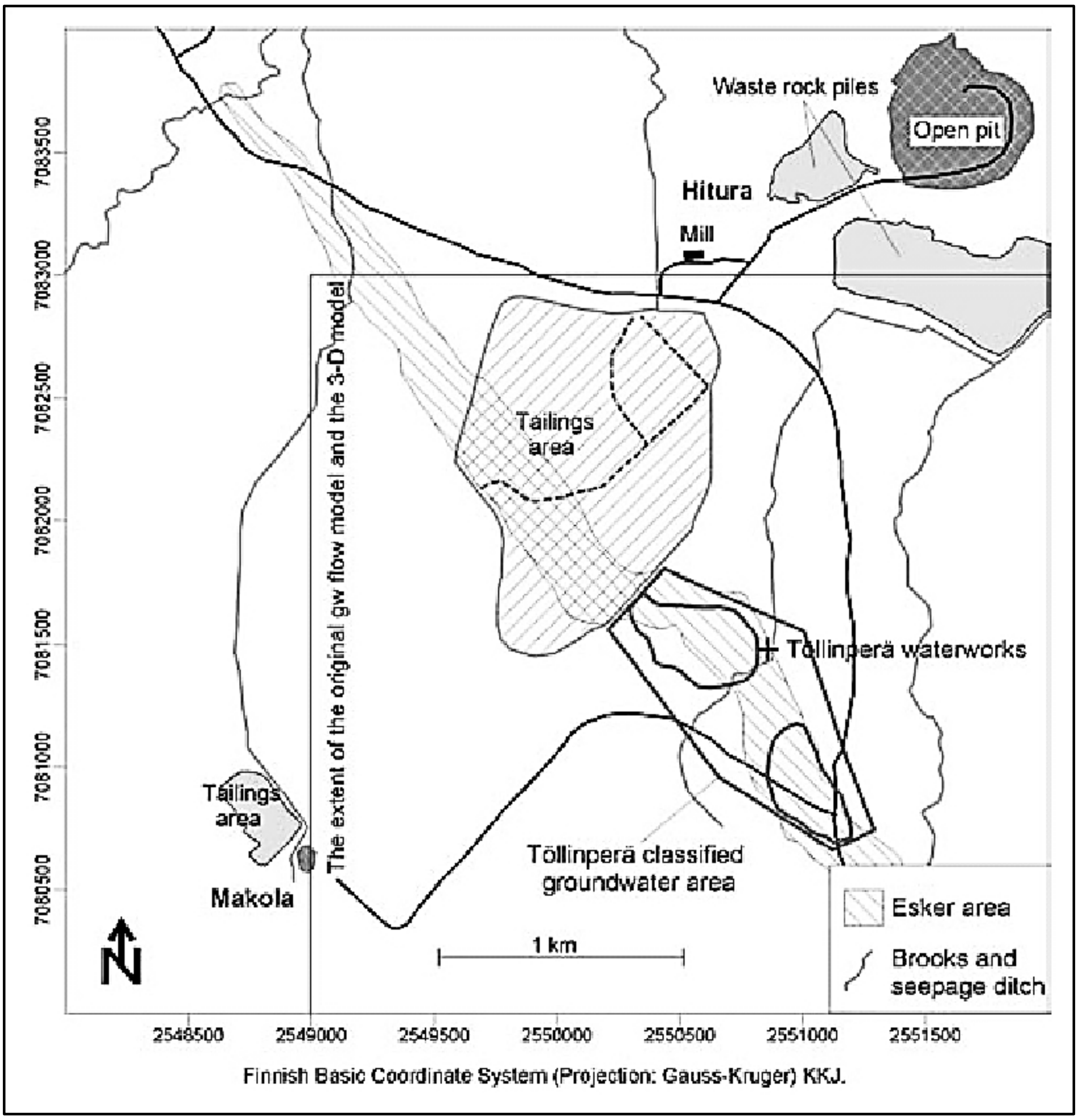


Figure 20-1: The Weichselian esker system with the Töllinperä groundwater area and groundwater recharge areas (inner boundaries) (Source: Artimo et al. 2004).

20.1.2 Kopsa Area

20.1.2.1 Groundwater Characteristics

The Applied Mining License Area (AMLA) has an approximate elevation of 110 masl. This compares to an elevation of 109 masl at Lake Levälampi and an approximate elevation of 75 masl at Lake Kortejärvi (part of the Kalajoki water course).

Two aquifers have been identified within the study area, namely the Lepola and Lahdekangos aquifers (Figure 20-2).

The soil in the Lepola aquifer area consists of 98.9% moraine and 1.1% peat with a soil layer thickness of between 2 m and 5 m. The overburden of the Kopsa property was deposited during and immediately after the end of the last glaciation. Only one outcrop occurs on the property, the remaining being buried beneath a shallow layer of till, soil and peat.

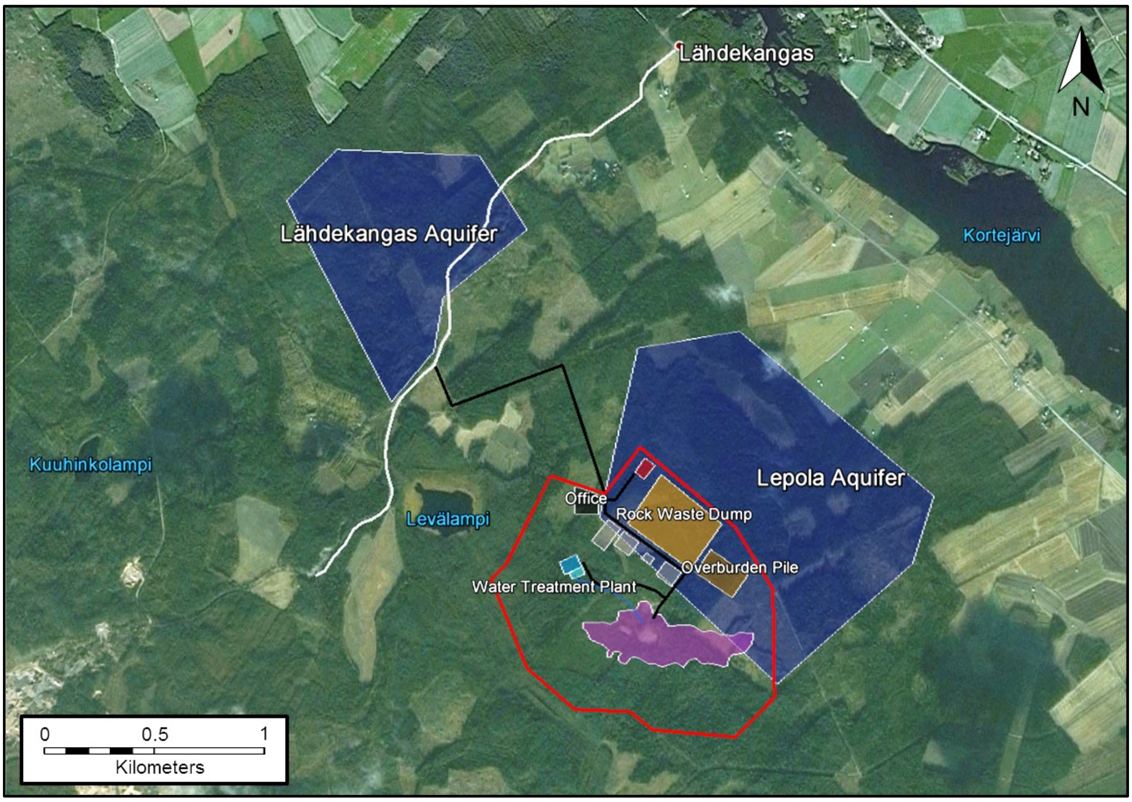


Figure 20-2: Overview of the Kopsa area* (Source: SRK, July 2013).

**Blue polygons: Approximate aquifer areas; Grey polygons: Sorting and storage facilities; Grey line: Road; Purple polygon: Mineralization area; Red polygon: Explosives storage; Red line: Applied Mining Licence Area*

There are three classifications for aquifers in Finland, Class I-III.

Class I: A groundwater area that is important for water supply i.e. a groundwater area from which 10, or more dwellings obtain their water.

Class II: A groundwater area suitable for water supply, but for the time being is not needed.

Class III: A groundwater area that needs further study to determine its feasibility as water supply.

The Lepola aquifer partially overlaps the AMLA as well as the southern corner of the future pit. It is currently categorised as a Class III aquifer, which indicates that it is unutilised, despite being a potentially suitable source. The aquifer will be removed from register in October 2013. Its potential suitability suggests that the hydraulic conductivity and specific yield of this aquifer are reasonably elevated. However, the limited aerial extent of the formation also probably means that the groundwater stored in the aquifer would become rapidly depleted were it to be dewatered. The proximity of this aquifer to the proposed pit means that it will almost certainly need to be dewatered both to limit the potential for high pore pressures behind the crest and to prevent large volumes of groundwater flowing into the pit and interrupting operations. Having highlighted the above, the magnitude of such impacts remains difficult to gauge, as the reviewers have been unable to obtain actual hydrogeological data on this formation.

The Lähdekangas Aquifer (Class I) supplies approximately 15 households with freshwater and is situated approximately 0.75 km from the north western edge of the AMLA. Although more remotely situated in relation to the pit than the Lepola aquifer, the evident importance of this aquifer as an existing water source for the local community means that the hydraulic connectivity through the geosphere between the pit and aquifer will have a significant bearing on the impact the mine has on this aquifer. The hydrogeological characteristics of this aquifer are not reported anywhere in the documents reviewed by SRK; however, the fact that it is used to supply local residents does suggest that the hydraulic conductivity and specific yield of this formation are quite elevated.

20.1.2.2 Surface Water Characteristics

The Kopsa site is located within the Kalajoki river basin near the Kalajoki River (~2 km). The river begins in Resijärvi, flowing north east to Haaparärvi. The river runs in a north westerly direction from Haapajärvi, passing the Kopsa and Hitura sites and from thence into the Baltic Sea. Flow in the river is regulated by four water dams that are located along its course. The Kortejärvi is regulated by a former mill dam and has an average depth of less than one metre. The lake is eutrophic due to agricultural activity.

A baseline study of surrounding water bodies was undertaken in 2007 - 2008 and monitored the Levälampi, Leväoja, Honkilampi and Myllyoja lakes. It also reviewed historical records of monthly data (flow rates and water levels) from measuring stations along the Kalajoki River nearest the Kopsa site (Haapajärvi and Oksava).

The on-going EIA baseline study is currently monitoring surface water quality. However, it has not been confirmed whether river stage monitoring will be included in the EIA baseline study. Hence, the monitoring programme may have to be expanded during the next stage of evaluation in order to create water balances for the mine catchment.

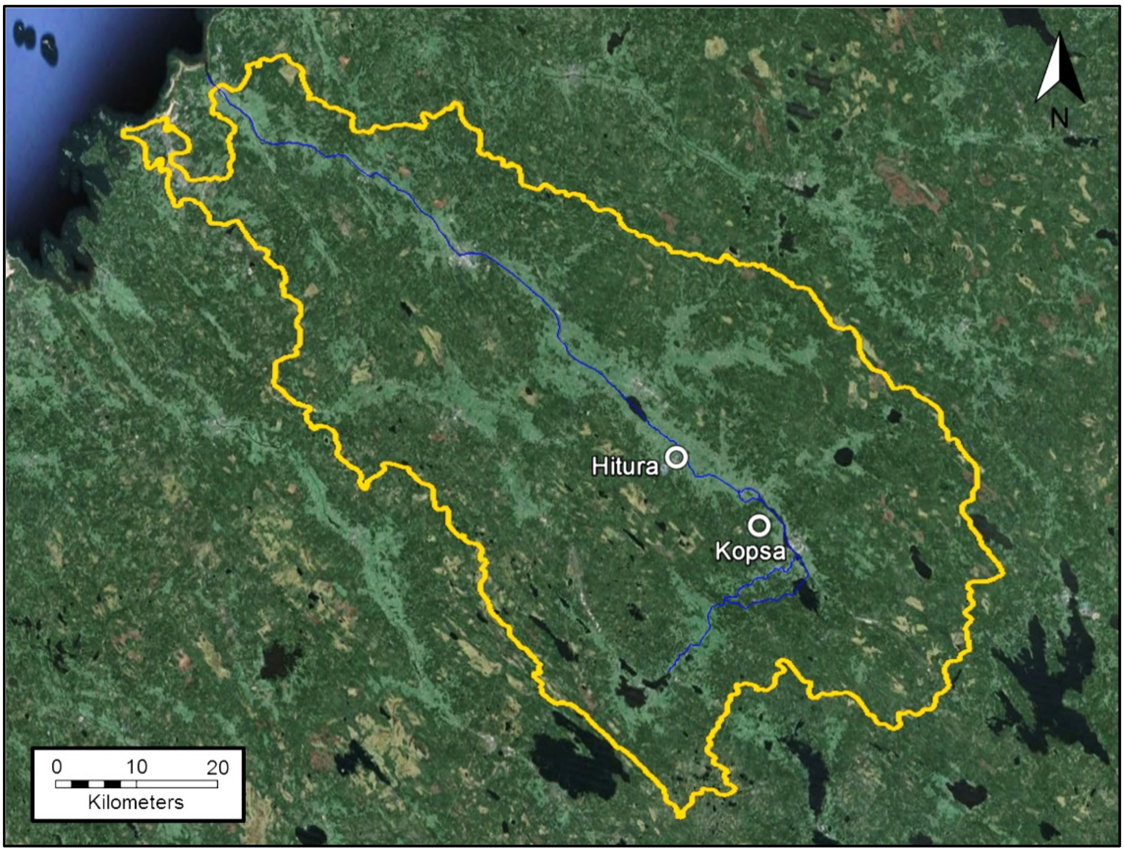


Figure 20-3: Kajaloki River Catchment Area (Source: SRK, July 2013).

20.1.3 Conceptual Surface and Groundwater Model for the Kopsa Site

20.1.3.1 Introduction

This section presents a conceptual understanding of the surface and groundwater systems at the Kopsa site for ambient (pre-mining) conditions (Figure 20-4), for conditions during mining (Figure 20-5) and for conditions after closure.

These models are based on a very limited data set and should therefore be considered indicative only. An important additional purpose of this section is to identify gaps in the present understanding of the system and use this information to design future field campaigns to capture required data.

20.1.3.2 Current (Ambient) Conditions

The groundwater flow regime is likely to flow out from the mineralization area pre- (and post) mining given that the area is slightly elevated compared to its surroundings. The groundwater regime may however be complex due to the presence of anthropogenic drainage networks and locally conductive sedimentary horizons usually found in eskers, kames and other coarse grained periglacial landforms. A site wide hydrogeological characterization will therefore be required to determine the distribution of hydraulic properties and to map spatial and temporal patterns in groundwater flow.

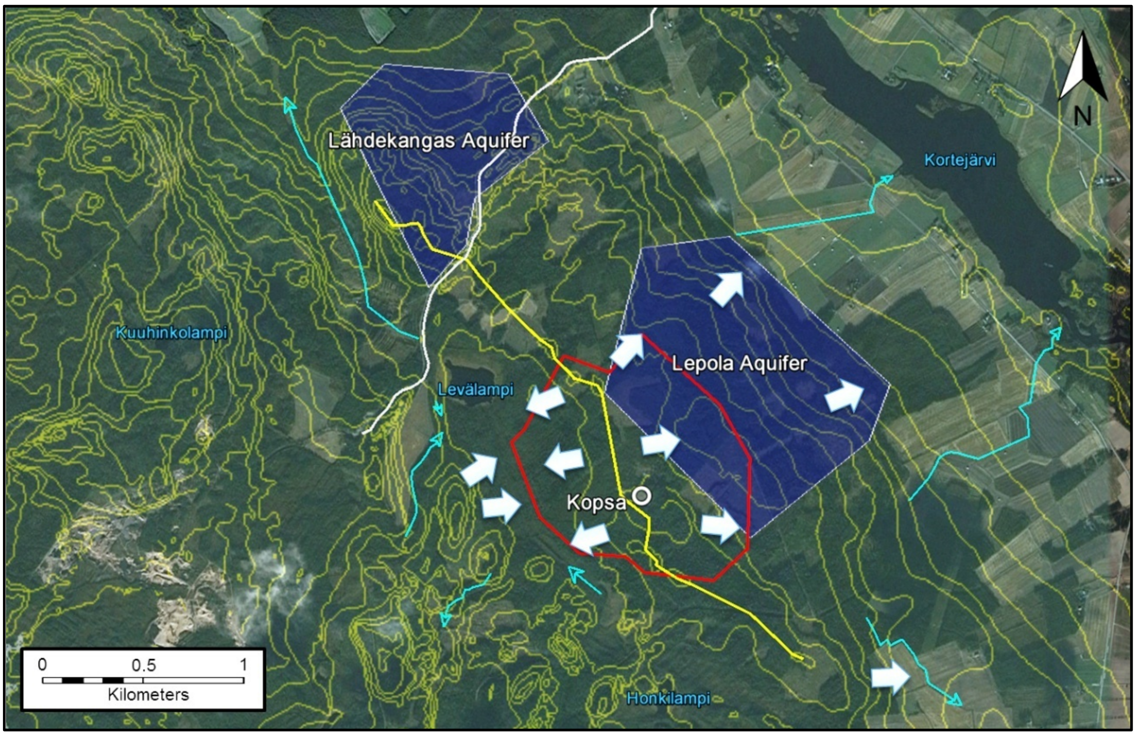


Figure 20-4: Ambient ground and surface water conditions* (Source: SRK, July 2013).

**Blue polygons: Approximate aquifer areas; Grey line: Road; Red line: Applied Mining Licence Area; Yellow line: Possible groundwater divider; White arrows: Possible groundwater flow direction; Turquoise arrows: Surface water flow directions*

20.1.4 During Mining Operations

Given its proximity to the future project site, the Lepola aquifer is likely to be affected by the dewatering required to operate the mine. A thick homogeneous permeable soil layer extending towards the Levälampi lake area could form a hydraulic connection to the open pit which will be approximately 120 m below ground level (~10 m below sea level). The depth that the pit will be developed to may mean that surface water bodies such as the Kalajoki River have the potential to drain towards the pit, especially if there is a good hydraulic connection, for example through fault structures. The mapping and hydrogeological characterisation of these structures in the bedrock as well as the overlying soil layers will play a key role in the hydrogeological understanding of the site and will require extensive field investigation in the next stage of technical evaluation.

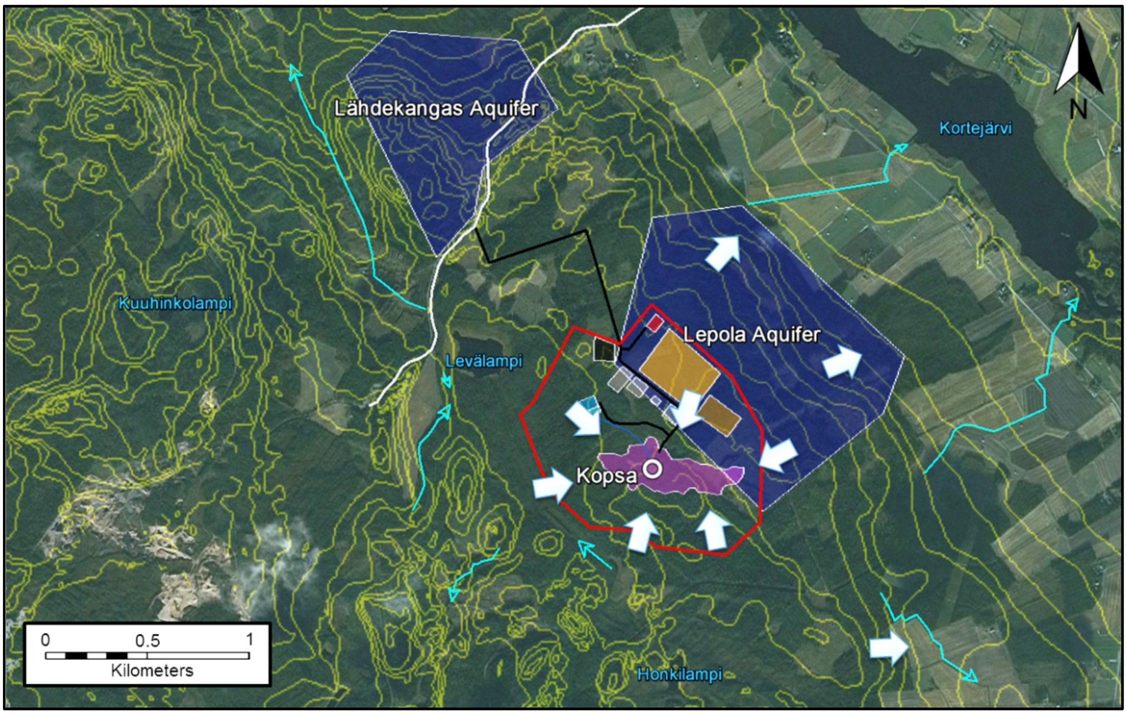


Figure 20-5: Overview of the Kopsa area with interpreted flow directions during mining* (Source: SRK, July 2013).

**Blue polygons: Approximate aquifer areas; Grey polygons: Sorting and storage facilities; Orange arrow: Possible but less probable groundwater flow direction; Purple polygon: Mineralization area; Red polygon: Explosives storage; Red line: Applied Mining Licence Area; Turquoise arrows: Surface water flow directions; White arrows: Possible groundwater flow direction.*

Post Mining Conditions

The behaviour of the surface and groundwater regimes around the Kopsa site after closure is expected to broadly reflect conditions as they existed before mining began. However, the flows local to the pit and mine complex are likely to be influenced both by the flooded pit and possibly by the partial removal or covering of sediments in the Lepola aquifer. The open pit closure plan will require, from a hydrogeological perspective, a detailed groundwater study to address potential impacts on local water regimes after mining has ceased.

20.1.4.1 Mine Water Management and Dewatering

Pit Dewatering and Pit Slope Depressurization

The open pit is expected to be in the order of 120 m deep with a total volume of 4.6 Mm^3. The final open pit area is estimated to be approximately 0.12 km^2.

The amount of dewatering required remains highly uncertain at this stage. However, a generic model has been constructed in MODFLOW to assess the potential groundwater inflow rate. This model assumes two layers: (i) a bedrock layer that has a bulk hydraulic conductivity of 10^{-7} m/s and (ii) an overlying 10 m thick layer of unconsolidated sediments that has a conductivity of 10^{-5} m/s. The model estimates a steady state inflow to the pit of 22 m^3/hr.

This dewatering rate has been estimated to help size the water management infrastructure and to provide indicative OPEX and CAPEX.

The unconsolidated sediment that overlie the bedrock is likely to be the largest source of water in the pit, particularly the nearby Lepola aquifer. It will be important to control the ingress of this water both to limit any interruption to mine operations and to keep the amount of water requiring treatment to an absolute minimum. Given the fairly elevated hydraulic conductivity that is assumed for this material, one solution is to install of ring of shallow vertical dewatering wells around the periphery of the pit where coarser sediments are identified. Additional vertical dewatering wells may also be required to intersect highly conductive faults in the bedrock, but at this stage the bedrock is assumed to be competent.

A separate geotechnical study will be required to optimise the pit slopes and it is possible that some depressurization of the pit walls will be necessary in order to stabilise the walls. The most common method for regulating pore pressures in the pit walls is through the installation of regularly spaced horizontal gravity drains from the pit benches. However, for the purposes of this study, it is assumed that the internal strength of the bedrock is such that only a very limited number of gravity drains will be required and that natural seepage to sumps located on the pit floor will be sufficient in most instances.

Pit water management also involves the control of surface water run-off both within the pit and from the catchments outside the pit. In-pit seepage will drain by gravity to the base where it will be captured in sumps and then removed to ponds located beyond the pit crest for treatment before being disposed of to the environment. The run-off from catchments located outside the pit is usually intercepted by drains installed around the periphery of the pit; this is designed to (i) prevent flooding of the pit following storm events i.e. limit interruption to operations in much the same way as vertical dewatering wells and to (ii) limit the amount of water requiring treatment. Since the future pit is located on elevated ground along the line of a mini-catchment divide, it is probable that the amount of run-off finding its way to the pit from outside will be very small and, as such there will be little or no need for interceptor ditches.

Water Quality Considerations

Reported geology and mineralogy suggests that arsenic (from arsenopyrite) and copper (from chalcopyrite) may be mobilised through contact with the pit walls, tailings and waste rock. Results from some initial geochemical tests on potential waste materials from the open pit confirm the mobility of arsenic and it is assumed water treatment will be required before discharge. This will need to consider both the operational and post-closure water treatment requirements as some residual drainage from the mine to the surrounding areas is likely given the higher topographic relief.

In addition, the early designs for the mine complex place certain mine infrastructure above the adjacent Lepola aquifer. There will be a risk that contaminants such as oils and degreasants will escape to the underlying aquifer unless suitable precautions are taken. This would likely include a combination of diversion ditches around the complex to limit the potential for run-off across the site, low permeability hard-standing, lined sumps and bunding.

Discharge Options

Initial scoping work identified three discharge options for water produced by the mine of which two were ultimately considered possible. The routes taken by the two preferred discharge pathways are shown in Figure 20-6 and described in more detail below.

Option 2 provides the shortest route from the mine to the nearest water course and will therefore cost the least to build. However, there is a case for reducing the impact on the Kortejärvi Lake and Kalajoki and this may be provided by Option 1, which will carry discharges to the neighbouring sub-catchment ~1 km from the site.

Both of these options are subject for further technical evaluation to determine maximum discharge, environmental and social risks involved as well as appropriate design and mitigation. The on-going EIA study will be able to provide some necessary baseline information that will support further evaluation of environmental impact.

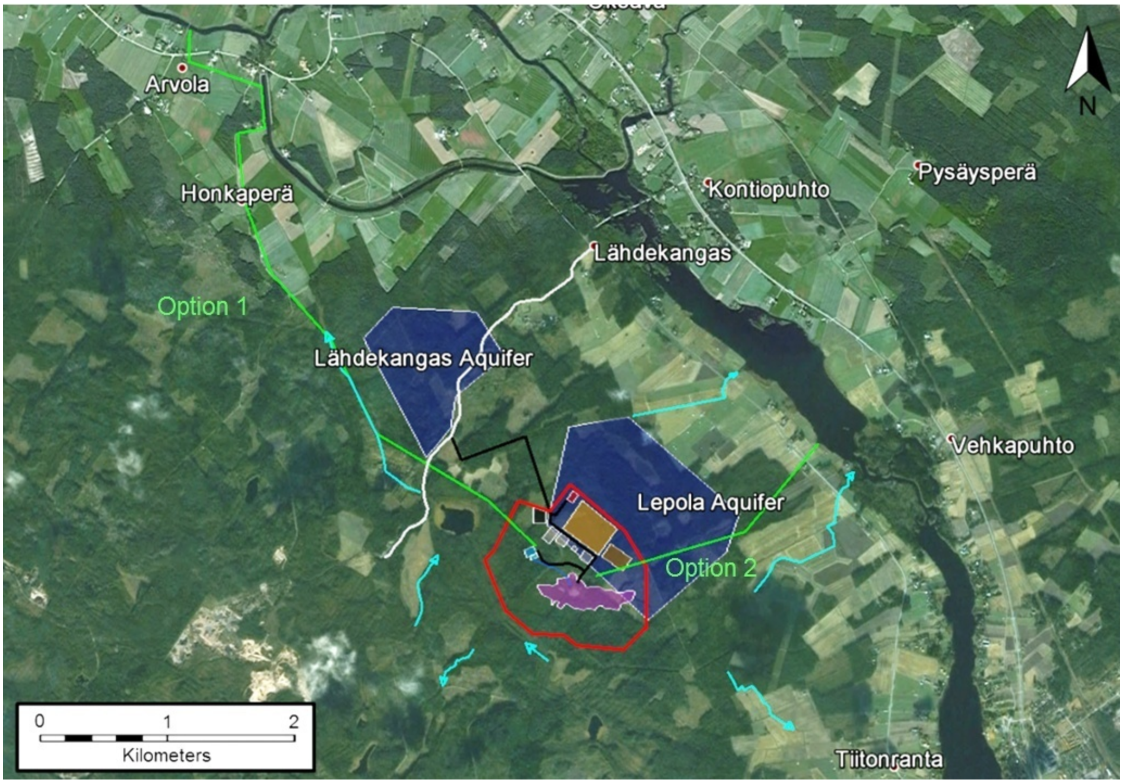


Figure 20-6: Overview of the Kopsa area with alternative discharge options* (Source: SRK, July 2013).

**Red line: Applied Mining Licence Area; Blue polygons: Approximate aquifer areas; Turquoise arrows: Surface water flow directions; White lines: Possible discharge pathways; Purple polygon: Mineralization area; Gray line: Existing road; Green line: Discharge pathway options*

20.1.5 Gap Analysis

SRK has undertaken a basic 'gap analysis' of the data that has been provided by the Company. Table 20-1 summarises our key findings, which includes all elements that would ordinarily contribute to the development of a robust conceptual understanding of the surface and groundwater regimes and to the design of a mine water management system that satisfies the requirements of a FS.

Table 20-1: Gap analysis of surface and groundwater data for the future FS

Study Area	Existing Data	Data and Information Gaps
Hitura: TMF Area	Soil characterization for the optional expansion of the TMF has been characterised in a technical study. Groundwater in terms of water quality is currently being monitored as well as monitoring of seepage recirculation. Groundwater monitoring for remediation purposes is on-going.	Expansion of current monitoring network Shallow groundwater levels, both spatial and temporal Hydraulic characteristics of the soil and near surface sediments under and immediately adjacent to the TMF footprint Seepage estimates to assist with the design of the TMF lining system
Hitura: Point of discharge Kalajoki River	Kalajoki river baseline data of historical river stages,	Recent river stage and water quality data.
Kopsa: The pit and mine complex	Some drillhole water quality data Geological drillhole assays from exploration activities	Hydraulic conductivity, storativity and hydraulic connectivity in the superficial sediments and in the underlying bedrock The distribution, frequency, orientation and hydraulic characteristics of faulting in the bedrock Conceptual surface and groundwater model of the mine site Numerical groundwater model to assist with the pit dewatering design, pore pressure control and to assess impacts on nearby receptors Detailed global mine water balance Preliminary designs for mine water infrastructure
Surroundings of the Kopsa deposit: including the Lepola and Lähdekangas aquifers	Surface water baseline study comprising of a desktop review and water quality sampling in 2008 that characterises surrounding surface water bodies. Historical groundwater quality and levels of the Lepola Aquifer Hydrocensus study to identify current water usage and water quality in the Lähdekangas Aquifer A surface and groundwater monitoring programme for the purposes of the EIA is currently being developed.	Hydrogeological characteristics of aquifers Near site sub catchment areas, surface and groundwater flow directions and flow rates.

20.1.6 Additional Surface and Groundwater Studies

The gap analysis has identified a large number of important omissions from the present study that will need to be addressed if the Kopsa Project is to be raised to feasibility study level.

Additional drillholes will be necessary to further support hydrogeological studies. It is SRK's opinion that the hydrogeological field campaign can be performed in partial collaboration with the geotechnical study to develop a joint drilling and testing plan, principally to assist with the hydraulic characterisation of the pit walls. This can be achieved through packer testing and flow logging in the geotechnical holes to quantify fault and fracture hydraulic conductivities followed by the installation of vibrating wire transducers at selected depths to establish spatial and temporal variations in pore pressure. This information should be supported by studies to look at the distribution, frequency, orientation and fill characteristics of faults and joints in the bedrock.

However, additional wells and pumping tests will also be required in the superficial material around the Hitura TMF, the Kopsa Pit and in the nearby aquifers to assess the impact that these more conductive materials are likely to have (a) on future pit inflows, (b) to help estimate the seepage potential from the base of the Hitura TMF and (c) ascertain the extent of hydraulic connectivity between mine site, aquifer and local surface water bodies. A selection of these holes should also be used to expand the current groundwater monitoring network, which should be upgraded to ensure that seasonal variations in groundwater level and groundwater quality are captured at the Kopsa and Hitura sites.

Baseline data collection for the EIA is current and expected to supplement the needs of the parallel FS study. However, it is important to ensure that the FS captures more information than currently exists about the surface water catchments around the Kopsa and Hitura sites through monitoring of flows from stream and river stages and the capturing of water quality data. This information should be used to model storm water run-off to the pit, the mine complex, the TMF and the waste rock dumps and ultimately to assist with the sizing and design of interceptor ditches and pit sumps (see Item 5 below).

Completion of the FS will also require the following tasks:

1. Development of robust conceptual surface and groundwater models for the Kopsa and Hitura sites;
2. Development of a numerical model of the Kopsa mine site to assist with the pit dewatering design, pore pressure control and to assess impacts on nearby receptors;
3. Refinement of the TMF water balance using the FS-level mine and TMF designs;
4. Incorporation of the TMF water balance in to a water balance for the entire mine complex (global balance) that includes pit inflows, process water streams, TMF losses, water recirculation together with water treatment and process make-up requirements; and
5. Preliminary designs for the mine water infrastructure such as dewatering wells, interceptor ditches, sumps, settlement ponds, pipes and pumps. This should include material quantities, power requirements and costs.

20.1.7 Estimated Costs

These costs fall in to three main categories, namely capital expenditure (CAPEX), sustaining capital and operational costs (OPEX). Indicative post closure monitoring costs are also estimated.

In terms of the present study, CAPEX has been used to cover the initial costs of acquiring and installing equipment, including for example drillholes, pumps and surface reticulation. The sustaining capital pertains to on-going costs associated with periodic repair and replacement of the existing infrastructure. OPEX relates to the on-going cost of running the existing infrastructure, which for water management purposes pertains mostly to energy consumption in respect of pumps and plant equipment and to a lesser extent to the costs associated with routine sampling of the monitoring network.

The costs that are summarised in this section are not based on supplier's quotes, but using industry standard valuations sourced from databases and SRK's own experience. For the purposes of this exercise, the accuracy of the costs should be considered commensurate with a PEA. They are also based on certain broad assumptions some of which may alter significantly as a result of further investigation during the FS stage.

20.1.7.1 CAPEX

SRK's estimate of the capital costs associated with installing the water management infrastructure at the Kopsa and Hitura sites are summarised in Table 20-2.

This study assumes that there is a requirement for a limited number of vertical wells around the periphery of the pit to dewater sections of higher permeability superficial material, particularly on the eastern side of the pit, and also horizontal gravity drains to help depressurise parts of the pit footwall and high wall.

Table 20-2: Project capital for water management infrastructure

Item	(1) Cost for LoM	Percentage Cost
Vertical Dewatering Wells	$150,000	15%
Horizontal Toe Drains	$300,000	30%
Sump Pumps	$150,000	15%
Surface Reticulation for Dewatering	$300,000	30%
Monitoring Wells	$100,000	10%
Total Cost	**$1,000,000**	**100%**

Notes:
(1). CAPEX does not include earthmoving and construction required to excavate in-pit sumps, drainage ditches around the pit, the TMF and the WRD or channels to dispose of surplus clean water to the environment. These are generally covered in the mining budget.

The principal capital outlay relates to the installation and equipping of the horizontal toe drains and the surface reticulation that together account for 60% of total expenditure.

In terms of timing of expenditure, it is expected that over 90% of the total capital outlay for LoM will occur in the first 2 years of operation.

20.1.7.2 Sustaining Capital

The sustaining capital costs associated with periodic repair and replacement of items within the water management infrastructure at the Kopsa Project are summarised in Table 20-3.

The annual sustaining capital for the Kopsa Project is USD 39,000.

The principal costs relate to the periodic replacement of drillhole and sump pumps; these account for approximately 60% of total expenditure.

Table 20-3: Sustaining capital for water management infrastructure

Item	Annual Cost	Percentage Cost
Vertical Dewatering Wells	$10,000	26%
Horizontal Toe Drains	$6,000	15%
Sump Pumps	$15,000	39%
Surface Reticulation for Dewatering	$6,000	15%
Monitoring Wells	$2,000	5%
Total Cost	**$39,000**	**100%**

20.1.7.3 Operating Costs

The estimated operating costs associated with running the existing infrastructure over LoM are summarised in Table 20-4.

Table 20-4: Operating costs for water management infrastructure

Item	Cost Type	(1) Annual Cost	Percentage Cost
Vertical Dewatering Wells	Power Usage (kWhrs)	$55,000	46%
In-Pit Sump Pump	Power Usage (kWhrs)	$50,000	42%
Water Monitoring	Lab and Materials	$15,000	12%
Total Cost	**-**	**$120,000**	**100%**

Notes:
(1). Labour costs to monitor, install and maintain the water management system are not considered in this section of the study. Such costs are generally covered in the mining budget.

The largest cost item will be the operation of the pumps in and around the Kopsa pit. The number of pumps and resulting costs should be considered indicative only at this stage. However, the estimates provided in Table 20-4 above suggest that energy consumption and routine maintenance of the pumps will very likely account for almost 90% of total operating costs, which is not untypical for an open pit operation. In terms of timing of expenditure, it is expected that operational costs will steadily increase over the operational life of the mine as the pit deepens and the amount of water and the head to pump against increases.

20.1.7.4 Post Closure Monitoring

Annual post closure water quality monitoring costs in Table 20-5 assume 5 surface and 5 groundwater points at each location, i.e. Hitura and Kopsa. These costs include other monitoring items such as water level readings and stream and river gauging. The costs assume that water quality is measured two times a year. Provision should be made for at least 5 years of post-closure monitoring, however the authorities may stipulate a longer period.

Table 20-5: Post closure water monitoring costs

Item	Hitura	Kopsa
Water Monitoring Points		
Surface	5	5
Groundwater	5	5
Frequency of Monitoring (Per Year)		
Surface	2	2
Groundwater	2	2
Lab Sampling Cost (Per Sample) = $258		
Equipment & Disbursement Costs =+15% of above		
Total Annual Lab and Kit Sampling Costs =$11,960		

Concerning labour costs, for each sampling round at Hitura and Kopsa provision should be made for 4 days including a site visit and reporting. Therefore annual labour will total 8 days per year.

20.1.8 Costs for Water Treatment

In terms of the potential water treatment requirements for Kopsa, a high density sludge precipitation style water treatment plant is recommended, where additional iron would be added to the inflowing waters to aid in the formation of an environmentally stable ferric arsenate. As the mine is significantly removed from the processing plant it is proposed that two water treatment plants are used, one at Kopsa and the other at Hitura.

20.1.8.1 Kopsa Pit

Based on a discharge from the open pit in the order of 11 m^3/hr, a plant designed to treat 15 m^3/hr will be required to treat pit water and runoff from the waste rock dump. The plant would cost in the order of €1 M and would require one operator to run during the day shift. The other operational costs are roughly estimated to be in the order of €0.15 / m^3 treated.

20.1.8.2 Hitura TMF

Based on a requirement to treat of the order of 5 m^3/hr of water from the TMF, a water treatment plant would cost in the order of €1 M and would require one operator to run during the day shift. The other operational costs are roughly estimated to be in the order of €0.15 / m^3 treated.

20.1.9 Closure

At cessation of mining operations the active water treatment plants should be replaced with suitable passive treatment schemes. Such schemes will minimise on-going operational costs as they will require minimal external input once operational. Typically the passive water treatment scheme will cost of the order of €0.4 M per scheme to create; one at each site. On-going costs will be of the order of 4 to 8 man days for maintenance purposes.

20.1.10 Conclusions

This report has presented the findings of the desktop study performed on the Kopsa and Hitura sites. Data has been compiled, reviewed, analysed and then, in the case of the Kopsa mine site used to construct a preliminary conceptual model. The model for Kopsa has been developed both to better understand the local surface and groundwater regimes operating at the site before, during and after mining and to assist in identifying gaps in the present data set that would need to be filled in order to complete a Feasibility Study.

Surface and groundwater in the project area is controlled by the topography, vegetation cover and geology. The relief is gentle with forest on the high ground and pasture in the valleys. The entire area has been subject to episodes of glaciation during the Quaternary which means that the fractured, crystalline bedrock is covered by a veneer of glacial till and glacial sands and gravels. This means that natural surface run-off is probably moderately sluggish unless there have been man-made drainage improvements. The groundwater system comprises two saturated units; namely a low hydraulic conductivity fractured crystalline bedrock overlain by a thin cover of glacial sediments with very variable hydraulic properties. For example, where there are thicker accumulations of sand and gravel, such as the Weichselian esker system under the Hitura site or the Lepola aquifer immediately to the east of the Kopsa mine, then hydraulic conductivity and storage are expected to be high. However, in areas where either glacial cover is thin, or where it is dominated by glacial till, then groundwater flows will be small.

During mining of the Kopsa pit, it is expected that the bulk of flow will come from the coarse grained glacial cover, in particular from the Lepola aquifer. The behaviour of surface and groundwater regimes at the Kopsa site after closure is expected to broadly reflect conditions as they existed before mining began. However, the flows local to the pit and mine complex are likely to be influenced both by the flooded pit and possibly by the partial removal or covering of sediments in the Lepola aquifer, an aquifer that given its proximity to the future project site is likely to be affected by the dewatering required to operate the mine.

The gap analysis has identified a need for an expansion of the surface and groundwater monitoring network and for hydrogeological characterization through pumping and packer testing of drillholes installed in the glacial cover material and in the bedrock. The hydrogeological characteristics and distribution of geological structures in the Kopsa area is not currently understood and this will also need to be addressed by further investigation of geotechnical and exploration holes.

The Hitura TMF may possibly be expanded with new storage cells which will modify the amount of discharge to the Kalajoki river and slightly alter the existing groundwater regime. The area has previously been subject to extensive hydrogeological mapping and new areas are expected to require less extensive testing than at the Kopsa site. Existing models will however need to be updated in order to re-assess remediation programmes and surrounding drawdown to the groundwater table.

The on-going EIA baseline study is currently monitoring surface water quality. However, it has not been confirmed whether river stage monitoring will be included in the EIA baseline study. Hence, the monitoring programme may have to be expanded during the next stage of evaluation in order to create water balances for the mine catchment.

Two discharge options have previously been identified and involve discharge to the Kalajoki river system. These options are subject for further technical evaluation to determine maximum discharge, environmental and social risks involved as well as appropriate design and mitigation and the on-going EIA study will be able to provide some necessary baseline information to support further technical evaluation.

Reported geology and mineralogy suggests that arsenic (from arsenopyrite) and copper (from chalcopyrite) may be mobilised through contact with the pit walls, tailings and waste rock. The quality from mine water discharges will therefore require further evaluation during the feasibility study to determine the extent of water treatment required. This will need to consider both the operational and post-closure water treatment requirements as some residual drainage from the mine to the surrounding areas is likely given the higher topographic relief. The open pit closure plan will therefore require, from a hydrogeological perspective, a detailed groundwater study to address potential impacts on local water regimes after mining has ceased.

20.1.11 Recommendations

A hydrogeological field programme is required to:

- Investigate surface water diversions with respect to mine water balance and choice of TMF option;
- Set up a permanent baseline monitoring programme in collaboration with the on-going EIA baseline study to support;
- Field campaign to support the dewatering system design and rock mechanical assessment; and
- Characterise groundwater conditions to support site selection plans for the tailings storage facility and waste rock dump locations.

Following completion of the field campaigns it will be necessary to:

- Review the final design of the TMF and WRD taking seepage and runoff management requirements into account;
- Construct a Site Wide Water Balance Model to further evaluate the need for mine water discharges;
- Construct numerical groundwater and geochemical models to support further assessments;

- Determine inflow rates to the open pit and contaminant transport post closure;
- Investigate the benefits of overburden dewatering to minimise mine water treatment;
- Develop a site wide water management plan;
- Confirm environmental impacts and permitting limits for mine water discharge;
- Develop a sustainable pit remediation plan with respect to open pit flow regime and water quality aspects; and
- Compile climate data statistics to further support designs for storm events and other contingencies.

20.2 Geochemistry

Acid Rock Drainage and Metal Leaching (ARDML) is probably the greatest long term environmental liability facing mining operations with sulphide deposits. ARDML arises when the in situ stable sulphide minerals are exposed to air and water through their excavation or ground disturbance. The resulting leachate can range from highly acidic to neutral effluents and is highly dependent on both the acid generating sulphide minerals and acid consuming carbonate minerals present in the deposit. In addition, acidic leachates can also mobilise metals/metalloids from surrounding minerals. Even neutral effluents can mobilise sufficient quantities of environmentally sensitive elements such as arsenic.

Although no ARDML geochemical assessment has been completed to date for the Project the reported geology and mineralogy suggests that:

- The acid generating minerals are dominated by arsenopyrite, chalcopyrite and pyrrhotite, with trace pyrite and other sulphides. Both arsenic, from arsenopyrite and copper, from chalcopyrite, are environmentally sensitive elements that could be mobilised.
- The host rocks will not offer much neutralisation potential but there is some reported acid consuming calcite mineralization within the mineralised zone.

Results from some initial geochemical tests on potential waste materials from the open pit confirm the mobility of arsenic from the materials and indicates that the majority of the rocks are uncertain as regards their acid generating potential. Further tests are still required to fully classify these materials.

Belvedere proposes that material will be processed at the existing concentrator at Hitura, which will be modified to process copper-gold bearing material. A number of options are considered for storing tailings at the existing TMF at Hitura. This existing facility has an historical ARDML issue in the form of a natural metal leaching environment and it is highly probable that copper-gold tailings will also be acid generating due to the sulphide minerals present. Similarly to waste rock, preliminary geochemical assessment has been completed.

Currently, due to the lack of geochemical characterisation, it is not possible to estimate the scale of potential environmental impacts from proposed mining and processing activities. A geochemical assessment programme has however been initiated by Belvedere to address this requirement. The programme will comprise both static characterisation, i.e. defining absolute classification of materials, and kinetic testing, i.e. defining the rates at which the predicted classification occurs. Justifiable mitigation controls can be attained from the test results.

20.2.1 Waste Legislation

While geochemical characterisation will aid in defining suitable mitigation controls, within Finish legislation there are also guidelines against which the Kopsa waste materials should be assessed. These guidelines are:

- *EU Directive 2006/21/EC – Management of Waste from Extractive Industries.* This Directive uses arbitrary classification to determine if waste is inert or non-inert based on its sulphur content. The Directive states that waste:
- With a sulphide sulphur content <0.1% can be classified as inert, so long as other criteria for potential contaminate release are met, i.e. there is no potential for environmental impacts from metal leaching.
- Materials with sulphide sulphur contents between 0.1% and 1% may be classified as inert so long as the ratio of acid buffering/consuming to acid generating potential is greater than 3 and other criteria for contaminate release can be met.
- Materials with sulphide sulphur content greater than 1% must be classified as non-inert.
- *Finnish Ministry of the Environment Guidelines for Extractive Waste Classification (2011).* This Directive specifies threshold values for selected metal(loid) concentrations in waste rock (Table 20-6).

Table 20-6: Finnish Ministry of the Environment guidelines for extractive waste classification

Parameter	Value (mg/kg)
As	5
Cd	1
Co	20
Cr	100
Cu	100
Hg	0.5
Pb	60
Ni	50
Sb	2
V	100
Zn	200

20.2.2 Recommendations

The majority of the waste at Kopsa will potentially exceed the arsenic and copper values in Table 20-6 due to the highly elevated occurrence of these elements within the deposit. In addition, arsenic and copper are also noted as being elevated in groundwater of the nearby Lepola aquifer and are therefore naturally mobilised from the deposit. Therefore for the purposes of this PEA, it has been assumed that treatment and containment facilities for both the waste rock and tailings will be required.

For the high-sulphur tailings it is also understood that sub-aqueous (in the open pit or underground workings) disposal is also being considered by Belvedere. This method, if operated correctly, is highly suited to controlling ARDML. In SRK's experience high sulphur wastes are best deposited as a cemented backfill deep within underground workings. The use of long term leaching tests will demonstrate the suitability of this disposal option.

As part of the feasibility study it is recommended that geochemical quantitative numerical predictions are also undertaken on all the waste and the pit lake that will form after closure. These predictions will aid in assessing the scale of potential impacts and confirm the suitability of selected mitigation controls. For this assessment, a full geochemical characterisation of all the materials will be required.

20.2.3 Costs for Water Treatment

In terms of the potential water treatment requirements for Kopsa, a high density sludge precipitation style water treatment plant is recommended, where additional iron would be added to the inflowing waters to aid in the formation of an environmentally stable ferric arsenate. As the mine is significantly removed from the processing plant it is proposed that two water treatment plants are used, one at Kopsa and the other at Hitura.

20.2.3.1 Kopsa Pit

Based on a discharge from the open pit in the order of 11 m^3/hr, a plant designed to treat 15 m^3/hr will be required to treat pit water and runoff from the waste rock dump. The plant would cost in the order of €1 M and would require one operator to run during the day shift. The other operational costs are roughly estimated to be in the order of €0.15 / m^3 treated.

20.2.3.2 Hitura TMF

Based on a requirement to treat of the order of 5 m^3/hr of water from the TMF, a water treatment plant would cost in the order of €1 M and would require one operator to run during the day shift. The other operational costs are roughly estimated to be in the order of €0.15 / m^3 treated.

20.2.4 Closure

At cessation of mining operations the active water treatment plants should be replaced with suitable passive treatment schemes. Such schemes will minimise on-going operational costs as they will require minimal external input once operational. Typically the passive water treatment scheme will cost of the order of €0.4 M per scheme to create; one at each site. On-going costs will be of the order of 4 to 8 man days for maintenance purposes.

20.2.5 Risks and Opportunities

The majority of the risks associated with the Project are associated with the extent and rate at which metals are mobilised from the waste. The more reactive the waste, the greater the risk.

It is fairly certain that a pit lake will form on closure of the Hitura mining operation. Therefore there may be an opportunity to either dispose the more reactive materials within the former open pit at Hitura before the lake forms, or in the flooded underground workings. The sub-aqueous environment that will be created within the former open pit and underground workings will aid in minimising the effect of the ARDML from the more reactive wastes.

20.3 Mine Waste Management (Tailings)

20.3.1 Introduction

This section presents the PEA level design for the Tailings Management Facility (TMF) as part of the PEA study. It derives a number of conceptual scenarios for tailings deposition by undertaking basic trade-off studies to identify a preferred alternative for storing and provide cost estimates. It is based on a tailings storage site located within or adjacent to the existing tailings facility. Technical Appendices A to C at the end of this report provide additional information on the study.

The scope of work for this study includes:

- desktop study based on available information;
- conceptual tailings site evaluation;
- conceptual dam design;
- conceptual water balance study;
- basic concepts for dam construction;
- conceptual slurry pipeline design; and
- indicative cost estimates sufficient to support the PEA (accuracy of up to ±50%).

20.3.2 Design Criteria

The selection of the type of tailings depends on several factors such as water availability, water management, dewatering capacity, site conditions, physical and geochemical properties, transport distance, climate, etc. Dry stacking was not considered for the Kopsa Project given the higher dewatering cost, climate, distance between the mill and the TMF. Paste tailings were also rejected on similar basis. Thickened tailings will require additional dewatering compared to conventional tailings slurry, and the higher density of the thickened tailings will result in higher pumping costs. Conventional tailings slurry will however have much larger quantities of excess water, thus requiring impoundments with larger storage capacities. The existing Nickel operation generated conventional tailings slurry.

Conventional tailings slurry requiring retention inside a paddock style impoundment was selected for the purpose of this PEA given the experience with such tailings by the existing operations and the facilities already in place. A trade off study should however be carried out at a later stage of the project for selecting the best option for the type of tailings.

It was assumed that the base of the TMF will be constructed with an impervious synthetic liner.

There is an opportunity to obtain project funding through the EU LIFE initiative. This potential funding calls for materials that is not necessarily from conventional sources, and/or may require additional handling due to the type of the material used. Although funding may be available, construction costs will likely be higher for meeting the requirements of the EU LIFE Initiative. The funding would consequently offset the incremental cost due to the alternative approach, and depending of the funding, may contribute to lower the overall cost of the TMF.

20.3.3 Design Parameters

A summary of the design criteria applied to the tailings management is provided in Table 20-7. The design criteria have been extracted from the existing project data developed by others and provided to SRK by the Client. National and international design standards have been used, and where appropriate, criteria assumed from SRK's experience and data gathered from other similar projects within the region have been used.

As mentioned elsewhere in this report, the material will likely be sorted at the mine site, thus reducing the tailings production by about 65% compared to the case with no sorting. The amount of tailings is estimated at 8.7 Mt without sorting and 3.0 Mt with sorting. The evaluation of the preferred option for tailings disposal is based on the case with sorting. The quantities and cost estimates were however prepared for both cases, to allow these to be assessed and compared in SRK's economic analysis.

Table 20-7: Summary Design Criteria

	Design Criteria	Unit	Value	Source / Comment
1	**Production Rates – No sorting**			
1.1	Tailings	Mt	8.7	SRK
1.2	Maximum annual tailings production	Mtpa	1.2	
2	**Production Rates – With sorting**			
2.1	Tailings	Mt	3.0	SRK
2.2	Maximum annual tailings production	Mtpa	0.4	
3	**Tailings Properties**			
3.1	Bulk dry density of settled tailings	t/m^3	1.50	Assumed (experience and/or adequate for PEA level studies)
3.2	Beach angle	%	0	
3.3	Specific Gravity of solids	-	2.73	
3.4	Porosity of settled tailings	%	45	
3.5	Solids content at discharge	%w/w	30	
3.6	Bulk density of tailings slurry	t/m^3	1.234	
4	**TMF Main Dam Properties at pipeline discharge**			
4.1	Freeboard requirement	m	3m	Assumed (experience)
4.2	Minimum crest width	m	10	Vehicle Access
4.3	Upstream slope	-	3:1 (H:V)	Slope Stability
4.4	Downstream slope	-	3:1 (H:V)	
6	**Sub-grade key trench of TMF Main dam**			
6.1	Width at bottom of key trench	m	3	Assumed (experience)
6.2	Depth	m	3	Assumed
6.3	Side Slope inclination	-	1:1 (H:V)	Assumed (experience)
7	**Design Storm Event**			
7.1	1-in-100-year annual exceedance probability (AEP), 24-hour	mm	80	Assumed based on SRK's experience within the region

The volume calculations for the deposited tailings are based on a horizontal tailings surface. Deposited tailings with a slope were discounted for the purpose of this PEA study. More detailed study should consider a tailings beach slope for the subsequent phases of the project.

The total tailings storage capacity required over life of mine using the above parameters is 5,817,220 m^3 for the case with no sorting, and 2,023,460 m^3 with sorting. The volume calculations are based on an in-place bulk dry density of 1500 kg/m^3. This volume excludes storage for excess water stored inside the impoundment.

The baseline study for the Kopsa area in 2008 lists average rainfall data between 1971 and 2000 and this data has been used as a part of the water balance. SRK used historical geographical data to define extreme rainfall events.

20.3.4 Design Assumptions for the TMF

The following assumptions have been applied for the proposed TMF:

- For the case with no sorting, a starter dam will be provided to store the tailings production for 1 year. The tailings will then be raised in a multi-phase approach. Subsequent studies will require analysis to determine the rate of rise of the dam;

- For the case with sorting, the tailings dam will be constructed to full capacity at the beginning of the mine life;
- The existing clarification ponds on the site are assumed to have sufficient capacity to accommodate the additional water from the proposed TMF;
- The land for the TMF will have been purchased and the cost of acquiring the land will not be considered as a part of the CAPEX for the TMF;
- Additional surface water allowance will be required if the TMF extends outside the footprint of the existing TMF; and
- All construction material for the proposed dam will be sourced from within a radius of 2 km from the TMF.

20.3.5 Design Assumptions for the Slurry Pipeline

SRK are not aware of any slurry test work undertaken for the tailings. SRK assumed that the slurry will be appropriate for pumping using centrifugal pumps along a pipeline and that the slurry will have non-settling Newtonian properties with acceptable flow velocities.

The conceptual design assumed the following design parameters:

- Operational for 24 hours a day for 300 days a year
- No infrastructure assets (services or roads) will need to be replaced.

20.3.6 Project Tools and Information

The tools used for analysis were the Global Mapper software tool and in-house software developed for application for Tailings storage and Slurry pipeline design.

The available information relevant to the project includes:

- Regional topographic maps:
  - ASTER DEM a product of METI and NASA; and
  - CAD drawing supplied from the Client.
  - Public domain information including aerial photography:
    - Google Earth - regional overview and aerial photography;
    - MapQuest open Street Map; and
    - World Topo Map.
- Projection KKJ, Zone 2, Finnish National Coordinate System, 1970 – 2003/2005

20.3.7 Ground Conditions

Belvedere Mining Oy has conducted a ground investigation in the area of the TMF.

The report states that the ground surface is mainly covered with peat with some silt, humus or sand. The peat thickness can be up to 1 m. The surface peat layer overlays a loose moraine layer. Some organic soils are also present below the loose moraine layer.

Test wells in the till identified numerous rocks and boulders of approximately 1 m diameter, typical of glacial formations. The groundwater was very close to the surface. The measured saturated hydraulic conductivity varied between $4.6x10^{-9}$ and $1.64x10^{-7}$ m/s.

The sub soil in the TMF area was found to be 0.5 m thick and consisted of a fine grained soil with probable low saturated hydraulic conductivity.

The existing TMF has intercepted an esker, thus indicating the presence of granular zones within the overburden in the area of the TMF.

20.3.8 TMF Option Selection Process

20.3.8.1 Existing Site

The area within and around the site of the existing Nickel TMF at Hitura was selected by the Client for storing the tailings produced by the Kopsa Project. The TMF is located approximately 12.5 km from the proposed mine site. The Hitura site can accommodate two options namely:

- Existing Nickel TMF; and
- Land to the south of the existing Nickel TMF.

The existing tailings disposal area is approximately 110 ha and has been constructed using embankments of local fine-grained till that rises approximately 30 m above the ground level. The existing TMF consists of four separate tailings impoundments. Two settling ponds are located on the north eastern section of the site for the process water as shown in Figure 20-7. The mine was operational for approximately 36 years and produced 12 Mt of tailings. The existing facility is surrounded by a seepage collection ditch. The two main lagoons are separated by a 10 m wide crest containing a rock fill embankment roadway. The land to the south of the existing tailings dam is currently woodland and a brook is located to the east of the site.

It is assumed that the previous operation used perimeter deposition using a series of spigots giving way to a beach slope of about 2%.

20.3.8.2 Initial TMF Option Selection

The option selection process used available data with the aim of evaluating economically viable options offering sufficient storage capacity with minimal impact upon the local environment. Relevant maps information were imported into Global Mapper software which were used to outline potential TMF options with sufficient storage capacity in the area of scope.

As mentioned earlier, the assessment of the preferred option is based on the case with no sorting of the material as it covers a greater range in terms of costs and potential impacts.

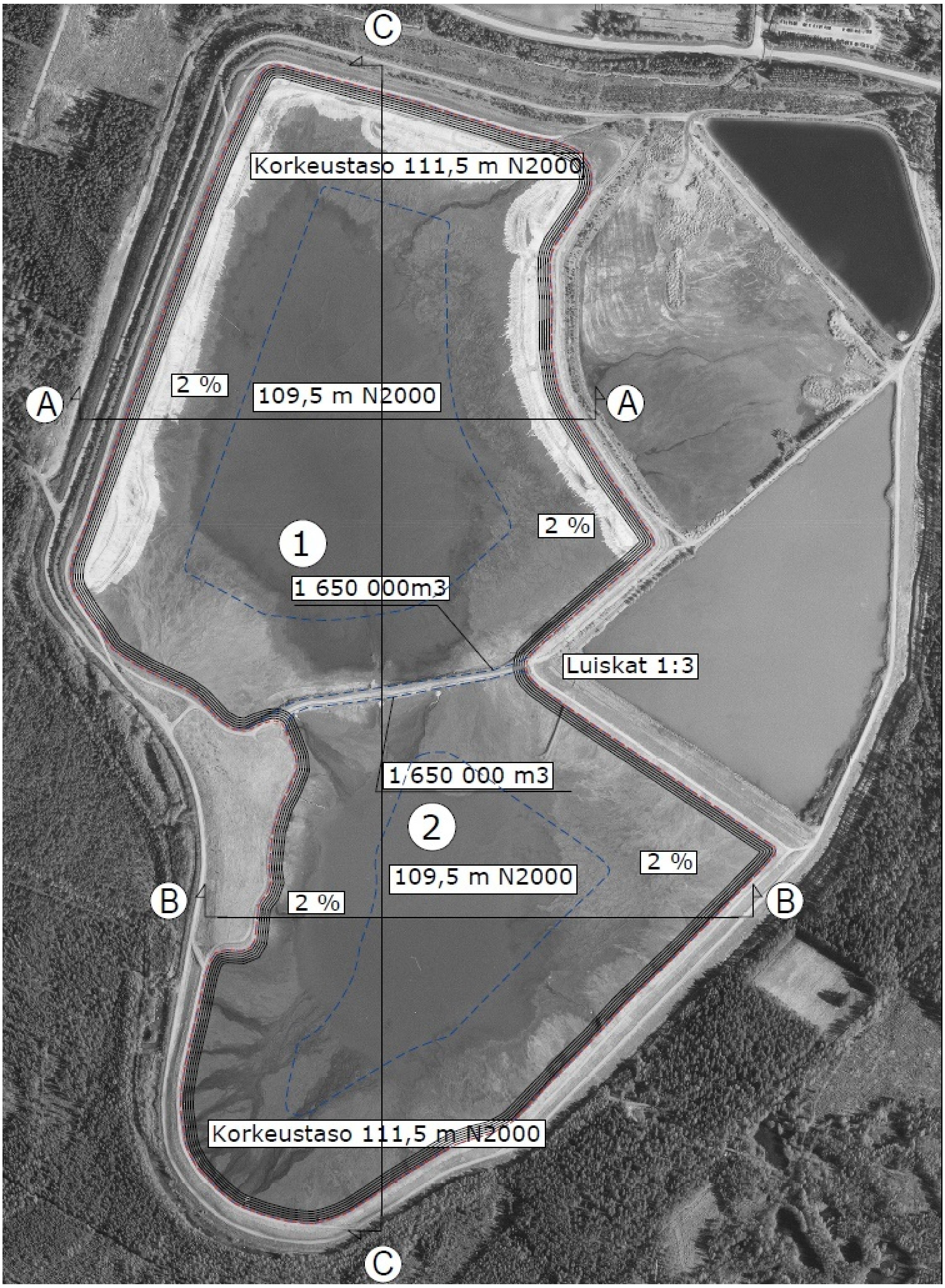


Figure 20-7: Existing tailings site (Client supplied information created by Finnish Consulting group) (Source: SRK, July 2013).

An indicative schematic cross section showing the key features of the likely perimeter dam design configuration is included in Figure 20-8.

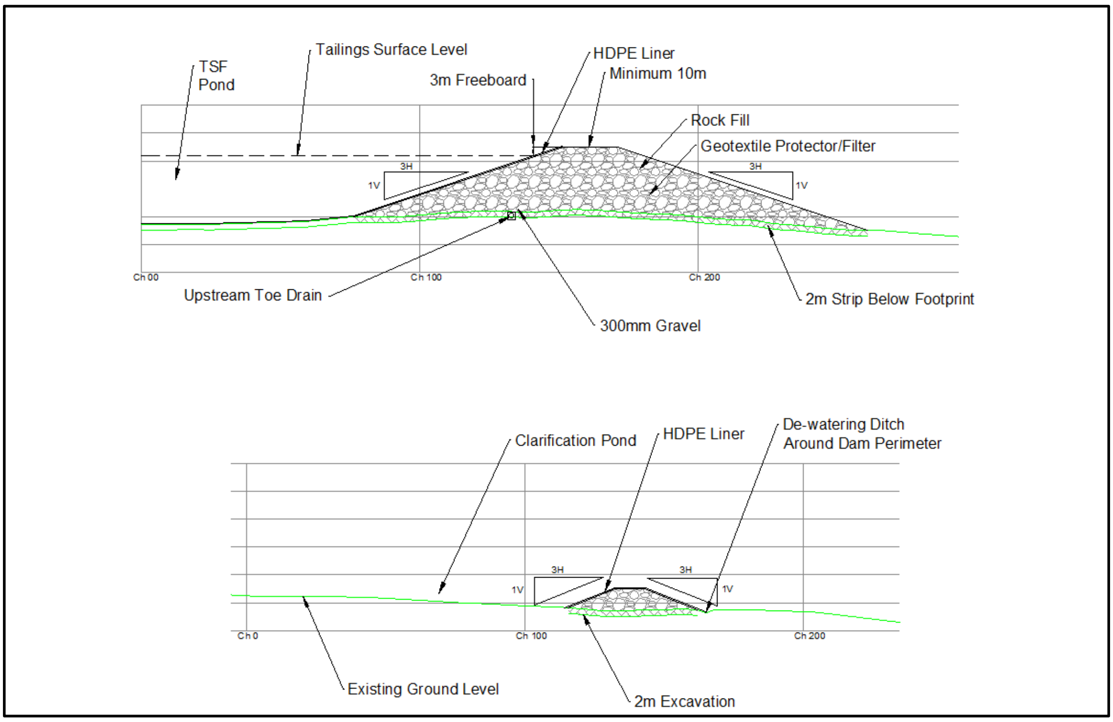


Figure 20-8: Schematic Cross Section Detailing Likely TMF Perimeter Dam and Clarification Pond Dam Design Constraints (Source: SRK, July 2013).

20.3.9 TMF Option Development

The selection process for the options was undertaken by initial analysis of volumetric, environmental and social factors. A trade-off study was undertaken to ascertain the suitability of the options in terms of available storage capacity and required footprint.

Based upon the initial findings, four TMF options as shown in Figure 20-9 were developed for consideration; the options are as follows:

- Option 1 – Raising the existing tailings lagoon 1 and 2 but keeping the existing roadway;
- Option 2 – New TMF to the south of the existing tailings facility;
- Option 3 – New TMF over the existing lagoons, removing the existing roadway; and
- Option 4 – New TMF over the existing tailings lagoon and the available land to the south.

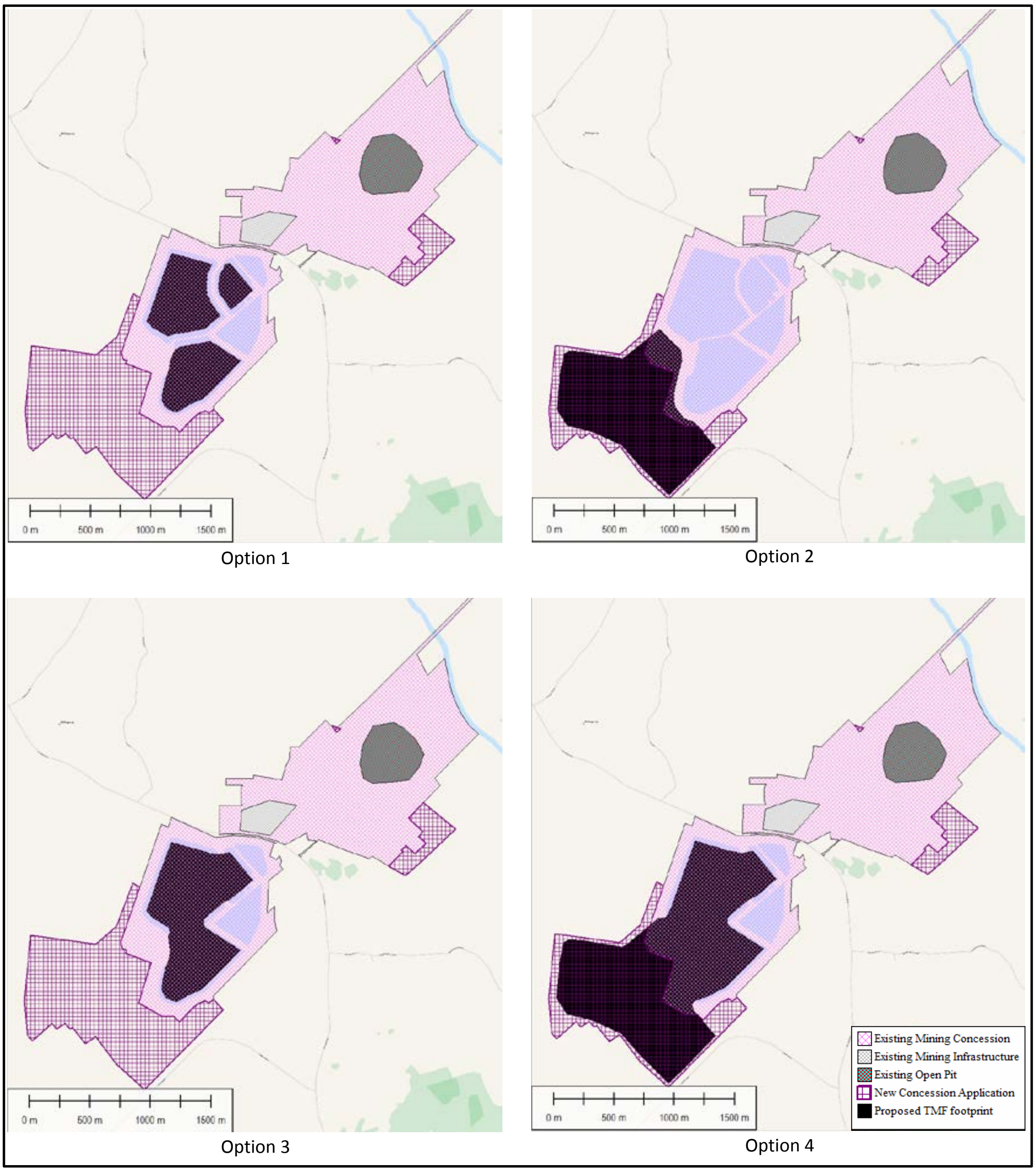


Figure 20-9: TMF Site Option Locations (Source: SRK, July 2013).

For each of the selected options, the following assessment tasks were undertaken:

- Determination of outline TMF option layout:
  - TMF perimeter dam layout, height and required fill volume;
  - Stored tailings volume and surface area;
  - Simple water balance calculations were undertaken to determine required water storage capacity for clarification; capacity sufficient to retain the water volume captured by the TMF area for a 1:100 year storm event occurring over a 24 hour period; estimated at 80 mm; and
  - Conceptual surface water management layout for the extension of the site.

- The environmental and socio-economic impacts of the proposed TMF options considered based on the following factors:
  - current land use – including stakeholders;
  - proximity to sensitive environments and heritage resources;
  - likely impact on surface water and groundwater;
  - visual impact;
  - disruption of existing transport routes;
  - public safety and environmental impact of a potential dam failure at the TMF; and
  - ground condition has not been considered in terms of geology or ground stability.
- Indicative cost estimate for the TMF:
  - CAPEX:
    - TMF starter perimeter dam construction;
    - Surface water management scheme construction; and
    - Closure costs.
  - OPEX:
    - TMF perimeter dam construction above the starter dam; and
    - Maintenance of the surface water management scheme.

A summary of options, size, construction requirements, fill requirements and outline cost assessments for the different tailings types considered is included in Table 20-8 for the case with no sorting and in Table 20-9 for the case with sorting (base case).

20.3.10 Cost Considerations

In the case with no sorting, the maximum volume of tailings material will not be reached until year 6 therefore the crest level of the proposed option can be lower in the early stages of the mine life. The construction cost for the dam has been spread over the life of mine. The case based on sorting would involve a single construction stage at the beginning of the mine life and will have a lower CAPEX given the smaller size of the TMF.

Exact information for achieving the EU LIFE funding is currently not fully known. It is understood that approximately €5M of funding would be available. It is believed that the additional requirements to achieve the funding may result in higher costs due to the additional requirements. Based on the current information the value of the funding has been omitted for the current assessment.

Table 20-8: TMF Option Summary, no sorting

Option Name			Option 1		Option 2		Option 3		Option 4	
			Starter	Final	Starter	Final	Starter	Final	Starter	Final
Tailings: Slurry	Surface area required for TMF development	km^2	0.269	0.561		0.80		0.638		1.671
	Maximum tailings elevation - TMF filled	m aOD	109.5	113.5, 114, 119	106.5	118.4	102.5	113	103	110
	Maximum dam height	m	15.5	25	11	22.4	8.5	19	9	16
	Volume of imported fill required for perimeter bunds	Mm^3	0.434	5.118	0.484	3.383	0.427	2.995	0.464	2.135
	Maximum Storage availability	Mm^3	0.924	5.82	0.862	5.82	0.924	5.82	0.88	5.82
	Ratio storage over dam volume		2.13	1.14	1.78	1.52	2.16	1.94	1.74	2.73
	Total expenditure (CAPEX & OPEX)	M Euro	**33.5**		**26.1**		**24.0**		**29.0**	

Note: Dam material assigned a unit cost of €3.92 /m^3 based on a source located within 2km from the TMF. aOD – above Ordinance Datum

Table 20-9: TMF Option Summary, with sorting

Sorting Option Name			Option 1	Option 2	Option 3	Option 4
Tailings: Slurry	Surface area required for TMF development	km^2	0.269	0.410	0.638	1.671
	Maximum tailings elevation - TMF filled	m aOD	115.1	117.1	106.1	105.3
	Maximum dam height	m	21.1	19.5	12.1	11.30
	Volume of imported fill required for perimeter bunds	Mm^3	1.23	1.667	0.946	0.906
	Maximum Storage availability	Mm^3	2.02	2.03	2.02	2.02
	Ratio storage over dam volume		1.64	1.218	2.14	2.22
	Total expenditure (CAPEX & OPEX)	M Euro	**12.31**	**13.0**	**12.41**	**21.69**

20.3.11 Other Considerations

The environmental and social factors relative to site selection were considered. The results of a qualitative comparison exercise are discussed further within Table 20-10.

The selection process has been restricted to the existing Nickel tailings and the land immediately to the south. Due to the fact that all four options are located within the same area, each option will have similar environmental and social constraints with varying degrees of impact.

Option 1 has the lowest qualitative score due to the fact that the proposed option would be located within the existing tailings site and the existing access road would be retained. Three separate paddocks would be required for the design – increasing the amount of material required for construction resulting in the lowest ratio of storage to fill volume. There could be transportation of windblown dust over greater distances at higher TMF elevations. Option 1 also requires a dam with the highest crest and will result in a negative visual impact.

Option 4 has the best storage to fill volume ratio but has the highest qualitative score due mainly to the increased footprint covering the total available land and resultant change of land use to mining from mainly forestry and the greater aesthetic impact.

The proposed TMF may require lining to minimise contamination from the proposed new tailings material. (Hitura's current environmental permit authorises construction of new lined cells, albeit to accommodate nickel tailings.) The constructability of a liner to contain the new tailings on top of the exiting tailings could be difficult due to the underlying soft tailings. Additionally, these conditions may induce settlements that could introduce risks for the integrity of the liner. Raising the perimeter dams would however not be an issue.

It is estimated that Option 2 would require 2.27 Mm^3 of fill material which is approximately half the volume of tailings to be contained. Option 2 requires the highest amount of fill material for the starter dam but the second lowest for the total dam material. From an environmental point of view, impacts associated with Option 2 are broadly similar to Option 4. For both options new tailings areas do not overly permeable moraine of the Weichselian esker system (Section 20.1); full impacts of the preferred option on this esker system and implications for current mitigation (groundwater dewatering pumps) should be investigated in the FS.

Each option will disrupt the existing transport routes as it is assumed that the construction material will be transported by trucks from the mine site on public roads. TMF's requiring minimal construction material will result in lower impact due to the reduced frequency of truck movements.

Table 20-10: Qualitative Comparison of TMF Options

Item	Criterion (Locality Name)	Weighting		Option 1	Option 2	Option 3	Option 4
1	Ground Stability	4	Ranking:	**3**	**3**	**3**	**3**
2	Current Land Use	4	Ranking:	**2**	**4**	**3**	**5**
3	Sensitive Environments (e.g. protected species, riverine habitats) / Heritage Resources	4	Ranking:	**1**	**2**	**1**	**2**
4	Surface / Ground Water	5	Ranking:	**3**	**5**	**3**	**5**
5	Visual/aesthetic impact	3	Ranking:	**3**	**4**	**3**	**5**
6	Public Nuisance (dust, local activities)	4	Ranking:	**2**	**4**	**3**	**4**
7	Disruption of existing / current transport routes (existing public roads, rail, footpaths, etc)	3	Ranking:	**2**	**2**	**3**	**3**
8	Public safety relating to the TMF side slope failure zone of influence	5	Ranking:	**4**	**2**	**4**	**3**
			Un-weighted Total	20	26	23	**20**
			Weighted Total	82	105	93	**120**

Explanation of ranking:

1 – Insignificant, 2 - Low Significance, 3 - Medium, 4 - High, 5 - Major

20.3.12 TMF Selection Summary and Selection

Based on the results shown in Table 20-8 and Table 20-10, constructability and considering environmental factors at a high level; Option 2 is the recommended option. Option 2 is believed to provide the best balance between construction requirements, storage availability and the likely environmental impact.

If favourable permitting and geotechnical conditions were obtained, there could be an opportunity to deposit the initial tailings production within the existing Nickel tailings facility while the new TMF is constructed. This has not been investigated as a part of this study. Additional work is however required to demonstrate the feasibility of placing the tailings over the existing ones. This could present an opportunity to reduce cost.

20.3.13 Water Balance

20.3.13.1 Water sources for consideration

A number of water sources need to be considered when assessing the water balance for the TMF. The following inputs and outputs were considered when assessing a water balance for the preferred TMF option, as shown in Table 20-11. Due to the preliminary stages of the project some sources have been disregarded as shown. Currently water is discharged from the TMF via a trench to the Kalajoki River. The capacity of the trench currently is unknown. It is currently proposed that all water is to be returned to the processing area with the option of discharging via the existing trench.

Table 20-11: Water Balance Sources and Considerations

Symbol	Description	Considered	Source
P_t	Precipitation inside the TMF site	Yes	Snow, rain
R_b	Runoff from the beach	No	Assumed no beach formed - water falling within dam captured within the precipitation
R_e	Runoff from the embankment	Yes	Intercepted by the surface water ditches along the outer perimeter
D	Diverted upgradient runoff	No	Not applicable full perimeter dam
E_b	Evaporation from the beach	No	Assumed no beach formed - only take evaporation from pool
E_p	Evaporation from the pool	Yes	
G_p	Groundwater seepage to or from the foundation materials	No	Insufficient information to quantify
G_t	Groundwater seepage to or from the groundwater	No	Assumed pond will be lined to minimise contamination to the groundwater
M	Mill water return	Yes	Based on water balance of TMF
T_w	Water used to convey the tailings to the reservoir for discharge	Yes	Water used as a vehicle to transport the tailings
S_t	Water stored within the tailings	Yes	Entrapped water based on the porosity of deposited tailings

Figure 20-10 indicates how the different sources of water interact with the TMF to allow a water balance to be derived for the option with no sorting. Figure 20-11 shows the same diagram with the quantities for sorting.

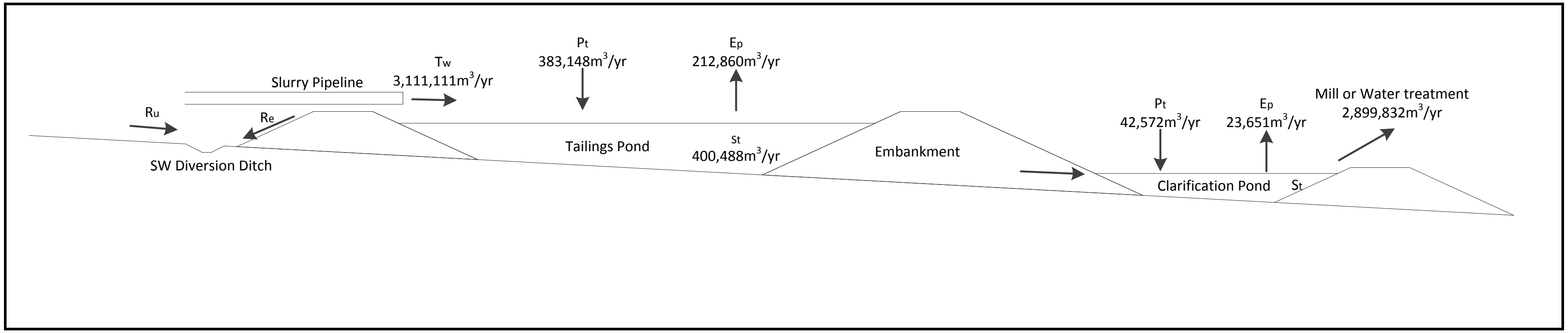


Figure 20-10: TMF Water Balance Diagram. (Source: SRK, August 2013).

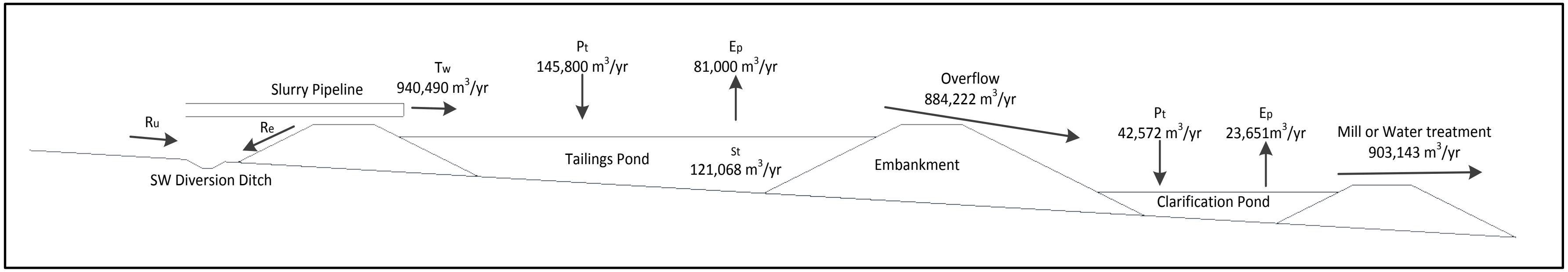


Figure 20-11: TMF Water Balance Diagram, With Sorting (Source: SRK, August 2013)

The water balance calculation indicates a probable deficit of water based on the assumptions presented herein. Additional fresh water would consequently be required.

20.3.13.2 Water Balance Summary

As shown previously, the TMF will receive about 2.70 Mm^3 of water per year from the tailings discharge on an assumed solid content of 30% for the case with no sorting. The volume of return water represents 2.51 Mm^3 per year which results in a water deficit of 0.19 Mm^3 per year that will need to be provided by a source of water.

For the case with sorting, the mill will discharge 0.94 Mm^3 of water while the return water will be 0.90 Mm^3 inflows, for a water deficit of 0.04 Mm^3 per year which will need to be obtained from another source.

The water deficit would remain about the same even if the solid content is increased or decreased. Leakage through the liner (loss of water) would consequently increase the quantity of water required to maintain the water balance at equilibrium.

A time dependant water balance will eventually be required at a later stage to define the water balance and to determine if additional water storage would be required over dry and low flow periods.

Detailed analysis of the water balance is shown in Appendix C.

20.3.14 Slurry Pipeline

20.3.14.1 Route

SRK understands that an existing pipeline SRK does not have information on buried services within the region and allowance for remedial works has been omitted for this study. Further investigations will be required in future studies.

The currently proposed pipeline route is shown in Figure 20-12.

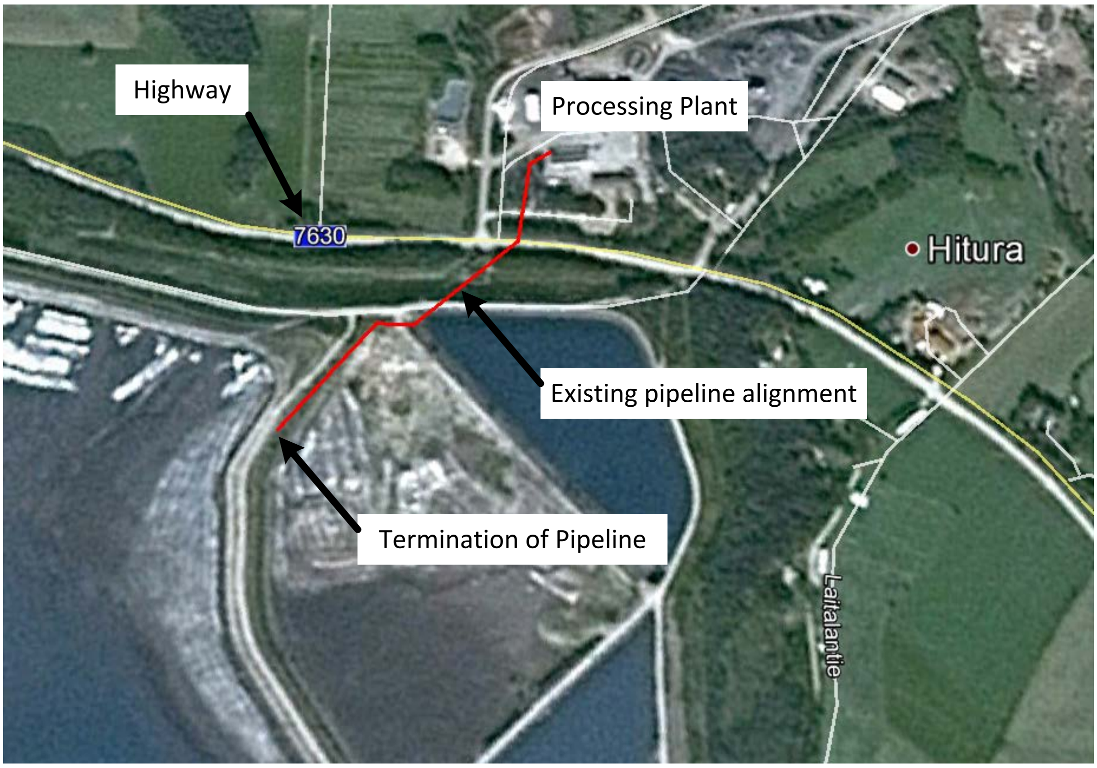


Figure 20-12: Proposed slurry pipeline route (Source: SRK, August 2013).

Based on the proposed route, the vertical profile was provided from the Client as shown on Figure 20-13.

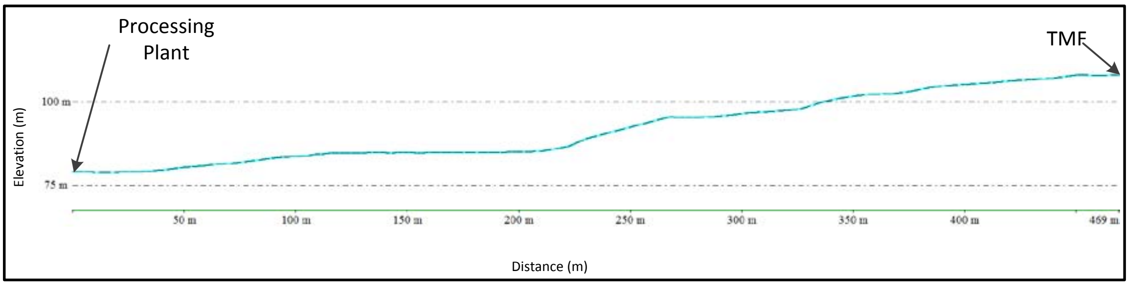


Figure 20-13: Proposed slurry pipeline elevation

The main elevation (approximate) properties along the proposed pipeline are:

- Processing site – 79 m;
- Tailings Management facility – 108 m;
- Lowest point – 79 m (at Processing site); and
- Highest point – 108 m (at the TMF).

The pipeline route will require refinement to minimise the energy required for slurry transport. Based on the topography it is believed that the water return pipe can be a gravity pipeline. An allowance has been made for the provision of a pump to aid in site activities to ensure that the water can be disposed form the TMF.

20.3.15 Pipeline Design

The hydraulic design of the slurry pipeline will require the rheology of the tailings material and more detailed topography. No test work has been undertaken to date for the tailings, and therefore, the properties as defined in Section 20.3.5 have been used for this conceptual design.

The conceptual pipeline design is based on steel pipe. There is an opportunity however to line the pipe with HDPE which would reduce frictional head losses. It is however conservative approach to use steel pipes as it will require pumps with higher capacities.

The details of the preliminary pipeline are included in Appendix A.

A 350 mm diameter pipe was selected for the slurry pipeline. It is estimated that a 45.7m head pump at the required flow rate.

A 350 mm diameter pipe was also selected for the return water pipeline. It is estimated that the system will be a gravity pipeline with the provision of two manholes to aid in maintenance and monitoring activities.

The design could probably be refined and provide an opportunity to decrease the pipeline diameter, thus a lower CAPEX. The smaller diameter would however increase the frictional head losses, and consequently require more powerful pumps or increasing the number of pumps (higher CAPEX). Such design refinement would be performed at a later stage of the project.

20.3.16 TMF Cost

Based on the conceptual design, the associated costs are shown in Appendix B.

The OPEX for the various options would be very similar due to the close proximity of the proposed TMF options.

It is estimated that the majority of the OPEX for the pipelines will result from the operational costs of the pumps.

20.3.17 Closure

The closure concept adopted for the TMF consists of the following items:

- Remove ponded and excess water within the impoundments;
- Breach the tailings dams as required;
- Installation of sumps to collect seepage water; and
- Placement of a soil cover to promote surface runoff and to support a sustainable vegetation cover.

The above closure concept assumes that the collected drainage water will be kept separate from surface runoff. The water that would have come in contact with the tailings will be directed to the water treatment plant for treatment. The clean surface runoff will be discharged with no treatment.

An indicative closure cost was estimated for the closure work described above and his summarised in Table 20-12 below for the case with no sorting (larger footprint) and for the case with sorting (smaller footprint).

Table 20-12: Cost estimate for closure, no sorting

Item	Quantity	Unit	Unit cost	Cost
Excess water removal	1	lump sum	€ 50,000.00	€ 50,000.00
Breach dam	60,000	m3	€ 2.50	€ 150,000.00
Sump for seepage water	1	lump sum	€ 50,000.00	€ 50,000.00
Soil cover	800,000	m2	€ 1.00	€ 800,000.00
Vegetation cover	800,000	m2	€ 0.25	€ 200,000.00
Instrumentation	1	lump sum	€ 50,000.00	€ 50,000.00
Subtotal				€ 1,300,000.00
Contingency	20%			€ 260,000.00
Engineering and procurement	5%			€ 65,000.00
Total				€ 1,625,000.00

Table 20-13: Cost estimate for closure, with sorting

Item	Quantity	Unit	Unit cost	Cost
Excess water removal	1	lump sum	€ 50,000.00	€ 50,000.00
Breach dam	40,000	m3	€ 2.50	€ 100,000.00
Sump for seepage water	1	lump sum	€ 50,000.00	€ 50,000.00
Soil cover	410,000	m2	€ 1.00	€ 410,000.00
Vegetation cover	410,000	m2	€ 0.25	€ 102,500.00
Instrumentation	1	lump sum	€ 50,000.00	€ 50,000.00
Subtotal				€ 762,500.00
Contingency	20%			€ 153,000.00
Engineering and procurement	5%			€ 38,000.00
Total				€ 953,500.00

20.3.18 Cost Summary - TMF

The following tables provide a summary of the costs associated with Option 2, the preferred option based on the assessment presented herein. The tables below include the cases with and without sorting. Most of the unit costs used for the cost estimates below are comparable to the ones used by the Client for their current operations and expansions.

Table 20-14: Estimated CAPEX for TMF construction, no sorting

Item	Quantity	Unit	Unit cost	Cost
Startup				
Demolition and removal from site - Wooded areas	80	ha	EUR 2,066.19	EUR 165,295
Trimming of excavated surfaces	800,000	m^2	EUR 0.81	EUR 649,026
Preparation of excavated surfaces	800,000	m^2	EUR 1.62	EUR 1,292,598
Granular base for liner	560,808	m^2	EUR 3.92	EUR 2,195,563
Liner	560,808	m^2	EUR 4.58	EUR 2,567,673
Dam construction (material & placement) - Stage 1	484,309	m^3	EUR 3.92	EUR 1,896,068
			Total	**EUR 8,766,223**
Post-startup				
Dam construction (material & placement) - Stage 2	362,385	m^3	EUR 3.92	EUR 1,418,738.00
Dam construction (material & placement) - Stage 3	362,385	m^3	EUR 3.92	EUR 1,418,738.00
Dam construction (material & placement) - Stage 4	724,771	m^3	EUR 3.92	EUR 2,837,477.00
Dam construction (material & placement) - Stage 5	724,771	m^3	EUR 3.92	EUR 2,837,477.00
Dam construction (material & placement) - Stage 6	724,771	m^3	EUR 3.92	EUR 2,837,477.00
			Total	**EUR 11,349,907**
			Grand total	**EUR 20,116,130**

Table 20-15: Estimated CAPEX for TMF construction, with sorting

Item	Quantity	Unit	Unit cost	Cost
Demolition and removal from site - Wooded areas	41	ha	EUR 2,066.00	EUR 84,714
Trimming of excavated surfaces	410,000	m^2	EUR 0.81	EUR 332,626
Preparation of excavated surfaces	410,000	m^2	EUR 1.62	EUR 662,456
Granular base for liner	270,000	m^2	EUR 3.92	EUR 1,057,050
Liner	270,000	m^2	EUR 4.58	EUR 1,236,202
Dam construction (material & placement)	1,655,054	m^3	EUR 3.92	EUR 6,479,536
			Total	**EUR 9,852,584**

Table 20-16: Cost summary, no sorting

Item	Cost
CAPEX	
TMF Construction Pre-operation	EUR 8,766,225
TMF Construction Operational	EUR 11,349,907
Water management	EUR 50,567
Water Return Pipe	EUR 81,353
Slurry Pipeline	EUR 239,490
Total CAPEX	**EUR 20,487,542**
OPEX	
Water management	EUR 22,930
Water Return Pipe	EUR 258,120
Slurry Pipeline	EUR 215,460
Total OPEX	**EUR 496,510**
Total Closure	**EUR 1,625,000**

Table 20-17: Cost summary, with sorting

Item		Cost
CAPEX		
TMF Construction Pre-operation		EUR 9,852,584
TMF Construction Operational		EUR 0
Water management		EUR 50,567
Water Return Pipe		EUR 81,353
Slurry Pipeline		EUR 239,490
	Total CAPEX	**EUR 10,223,994**
OPEX		
Water management		EUR 22,930
Water Return Pipe		EUR 258,120
Slurry Pipeline		EUR 215,460
	Total OPEX	**EUR 496,510**
	Total Closure	**EUR 953,500**

20.3.19 TMF Discussion and Recommendations

Option 2 with sorting was selected as the preferred option for the purpose of this PEA. The selected option consists of downstream dam construction method designed to contain conventional tailings slurry. This option will be re-assessed and compared to alternative options during the later stages of the project as additional information becomes available.

The tailings deposition considered for this PEA is based on discharging along the perimeter of the impoundment. The beach slope was assumed horizontal for this study but beach slope should be assessed and incorporated in the subsequent phases of the project. The key cost factor is transport of soil and rock fill materials between the mine site and the TMF location using road haulage.

Based on the findings stated in section 20.3.8; Option 2 located directly south of the existing Nickel tailings site is the preferred TMF site. Option 2 minimises the construction uncertainties as the option does not involve construction on the existing Nickel tailings. It is clear that sorting will reduce considerably the overall cost for tailings disposal as it will involve much lower quantities of tailings.

The project is still at an early stage and more technical work will be required for the subsequent phases of the project, in particular for:

- Key social and environmental factors;
- Site conditions (geotechnical, hydrogeological, hydrological, climate, seismic);
- Practicalities of utilising the existing Nickel TMF including the clarification pond;
- Tailings properties (physical, rheological and geochemical); and
- Regulatory requirements, etc.

The design of the TMF will also require interactions with the process and mining disciplines to assure proper integration of the TMF with the overall project. Future pipeline studies will require:

- Confirmation of the pipeline route;
- Consequences of a tailings pipeline failure between the mine site and the TMF;
- Consultation with product suppliers;
- Sources for water abstraction;
- The need for a water return pipe;
- Leak detection system; and
- Investigation of a choke station.

20.4 Waste rock and overburden dumps

20.4.1 Design

The waste rock and overburden dumps were designed as part of the pit design. The mining activities will generate about 4.2 Mt of waste rock and the output from sorting process will generate an additional 5.8 Mt of waste that will be also disposed at the waste rock dump.

Preliminary geochemical work on the waste rock material indicate that it will be acid generating, thus requiring the collection of the water coming in contact with the waste rock material. This will require the installation of an impervious liner at the base of the dump and all water originating from the waste rock dump will be collected and directed to the water treatment plant.

Although the data is limited at this stage, the overburden dump has been assumed to be benign and that it would not require a liner at the base. It was also assumed that the water coming in contact with the overburden dump would not need treatment and be suitable for discharge.

20.4.2 Closure

The closure of the waste rock dump will include a simple soil cover. It is assumed that the seepage water collected via the base liner will require treatment for post-closure. The soil cover will consist of 0.3 m of soils sufficient to support vegetation while providing stability against erosion.

It was assumed that most of the overburden dump would be consumed for post-closure work and that it would not need any additional work at post-closure.

20.4.3 Cost

20.4.3.1 Base line

The cost to haul and dump the waste rock material to the dump is included in the mining cost. The estimated cost to prepare the ground surface and place a synthetic liner is summarised in Table 20-18 for the case with no sorting and in Table 20-19 for the case with sorting. The sorting case will generate will more than double the amount of waste compared to the case with no sorting, thus essentially doubling the footprint of the waste rock dump.

Table 20-18: Estimated cost, base liner for waste rock dump, no sorting

Item	Quantity	Unit	Unit cost	Cost
Start-up				
Trimming of excavated surfaces	90,500	m^2	EUR 0.81	EUR 73,305
Preparation of excavated surfaces	90,500	m^2	EUR 1.62	EUR 146,610
Granular base for liner	90,500	m^2	EUR 3.92	EUR 354,760
Liner	90,500	m^2	EUR 4.58	EUR 414,490
			Total	**EUR 989,165**

Table 20-19: Estimated cost, base liner for waste rock dump, with sorting

Item	Quantity	Unit	Unit cost	Cost
Startup				
Trimming of excavated surfaces	181,000	m^2	EUR 0.81	EUR 146,610
Preparation of excavated surfaces	181,000	m^2	EUR 1.62	EUR 293,220
Granular base for liner	181,000	m^2	EUR 3.92	EUR 709,520
Liner	181,000	m^2	EUR 4.58	EUR 828,980
			Total	**EUR 1,978,330**

20.4.3.2 Closure

The estimated closure cost for the waste rock dump is summarised in Table 20-20 for the case with no sorting and in Table 20-21 with sorting.

Table 20-20: Estimated closure cost, waste rock dump, no sorting

Item	Quantity	Unit	Unit cost	Cost
Topsoil (0.3 m thick)	90,500	m^2	€1.25	€113,000.00
Vegetation cover	90,500	m^2	€0.25	€23,000.00
Instrumentation	1	Lump sum	€10,000.00	€10,000.00
Subtotal				€146,000.00
Contingency	40%			€58,000.00
Engineering and procurement	5%			€7,000.00
Total				€211,000.00

Table 20-21: Estimated closure cost, waste rock dump, with sorting

Item	Quantity	Unit	Unit cost	Cost
Topsoil (0.3 m thick)	181,000	m^2	€1.25	€226,000.00
Vegetation cover	181,000	m^2	€0.25	€45,000.00
Instrumentation	1	Lump sum	€10,000.00	€10,000.00
Subtotal				€281,000.00
Contingency	40%			€112,000.00
Engineering and procurement	5%			€14,000.00
Total				€407,000.00

20.5 Environmental and Social Assessment, Permitting and Management

The following Section includes discussion and comment on the environmental and social aspects pertaining to Belvedere's Hitura and Kopsa assets in Finland.

20.5.1 Scope of review

The status of the Project's primary authorisations is discussed in this Section. Salient environmental and social issues are reported and recommendations for future environmental and social studies are provided.

The following documents were reviewed;

- Updated Reserve and Resource Estimate of the Hitura Nickel Mine, National Instrument 43-101 Technical Report, Outotec (Finland), December 2012;
- Ecological baseline studies for the Project, Ahma Environment Ltd (Ahma), January 2008;
- Hitura environmental permit no. 66/10/1 dated 13 August 2010;
- Annual environmental report for 2012, Belvedere;
- Summary of permit status, 2013, Belvedere;
- Hitura waste management plan, Belvedere 2012;
- Hitura detailed closure plan, Belvedere, December 2012; and
- Kopsa progress presentations, Belvedere, 2012, 2013.

The following people were consulted during a site visit from 8 to 10 May 2013 and afterwards, during compilation of this Section;

- Markus Latvala - Environmental Manager, Belvedere Mining Oy ;
- Toby Strauss - Chief Operating Officer, Belvedere Resources Ltd;
- Jukka Nieminen - Chief Executive Officer, Belvedere Mining Oy;
- Jari Hietala - Managing Director, Ahma Insinoorit Oy (Ahma); and
- Heikki Miettunen - Process Manager, Belvedere Mining Oy.

Ahma carryied out environmental and social studies for the Project on behalf of Belvedere.

Pekka Tuomela from Pöyry Finland Oy Exploration and Mining Services in Rovaniemi advised SRK on the Finnish environmental authorisation process.

20.5.2 Project Setting

Kopsa: The proposed mine site is located on slightly elevated area on generally flat terrain east of the concession, which gently slopes from 110 mamsl near the mine to between 68,5 mamsl and 80,2 mamsl at the Kalajoki River bank approximately 2.1 km east of the concession (Figure 20-14). The terrain undulates immediately west of the concession where hillocks (120 mamsl to 130 mamsl) and rocky outcrops are dispersed in low lying bogs and swamps. The mine site is located on a drainage divide. Surface and groundwater in the eastern part of the mine is conveyed to the Kalajoki River and water in the western part of the mine is conveyed to bogs and swamps. Surface water features in the vicinity of the mine include drainage ditches and at least one stream just west of the mine, which flows in a north westerly direction. About 8 km upstream of the mine site is Lake Hautaperän. Lake Niini is 6 km south west of the mine and Lake Haagan is some 4 km south east of the mine. Numerous small ponds and creeks occur in the concession, the largest of which is Levä pond, a few 100 m north west of the concession. There are six springs approximately 0.7 km and 1 km west and east of the concession respectively. Two springs are in a natural state, i.e. the springs have not been artificially drained and at least one is used as a domestic water supply.

There are Class I and Class III groundwater aquifers in the vicinity of Kopsa (Figure 20-15). The Class I aquifer named Lähdekangas is roughly 1 km northwest of the mine, outside the concession. The aquifer is important for water supply because it provides 15 households with domestic water. Lepola is currently a Class III aquifer partly underlying the concession. The aquifer will be removed from the registry in October 2013. Class III aquifers require further study if they are to be used for water supply. Belvedere and the Finnish Environmental Institute (SYKE) monitor the quality of water in this aquifer. Belvedere reports 'poor' quality with elevated levels of cadmium, cobalt, mercury and arsenic, which are probably related to the mineralogical composition of the bedrock.

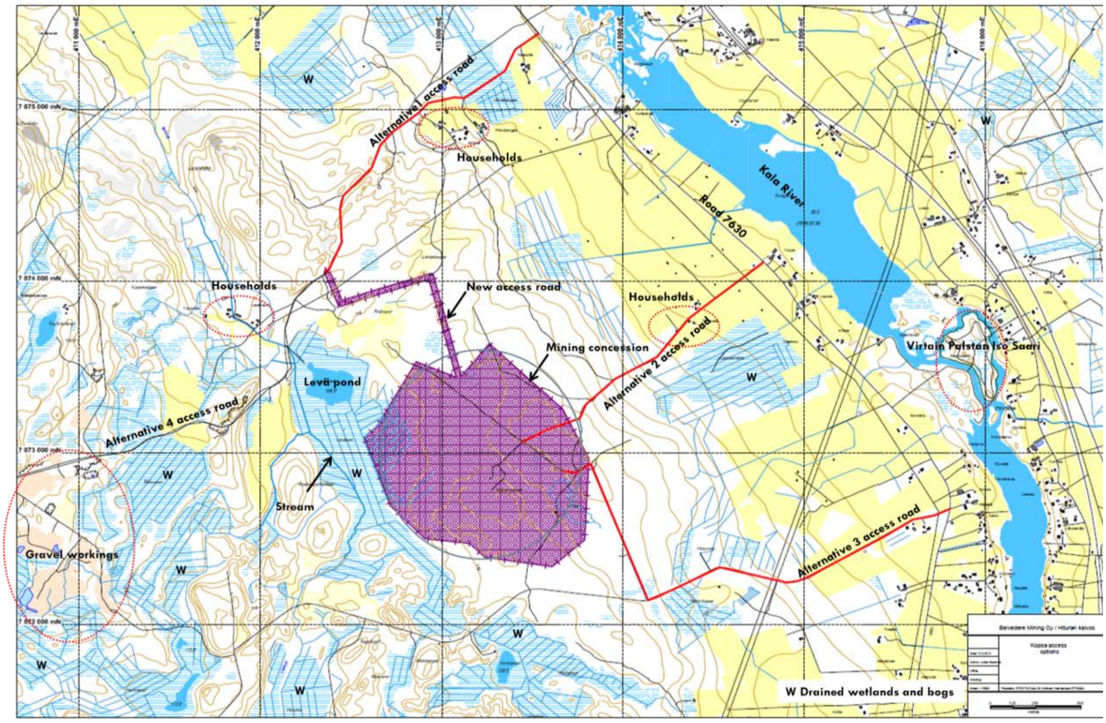


Figure 20-14: Proposed Kopsa mine site, mining concession and surroundings (Source: Belvedere, 2013)

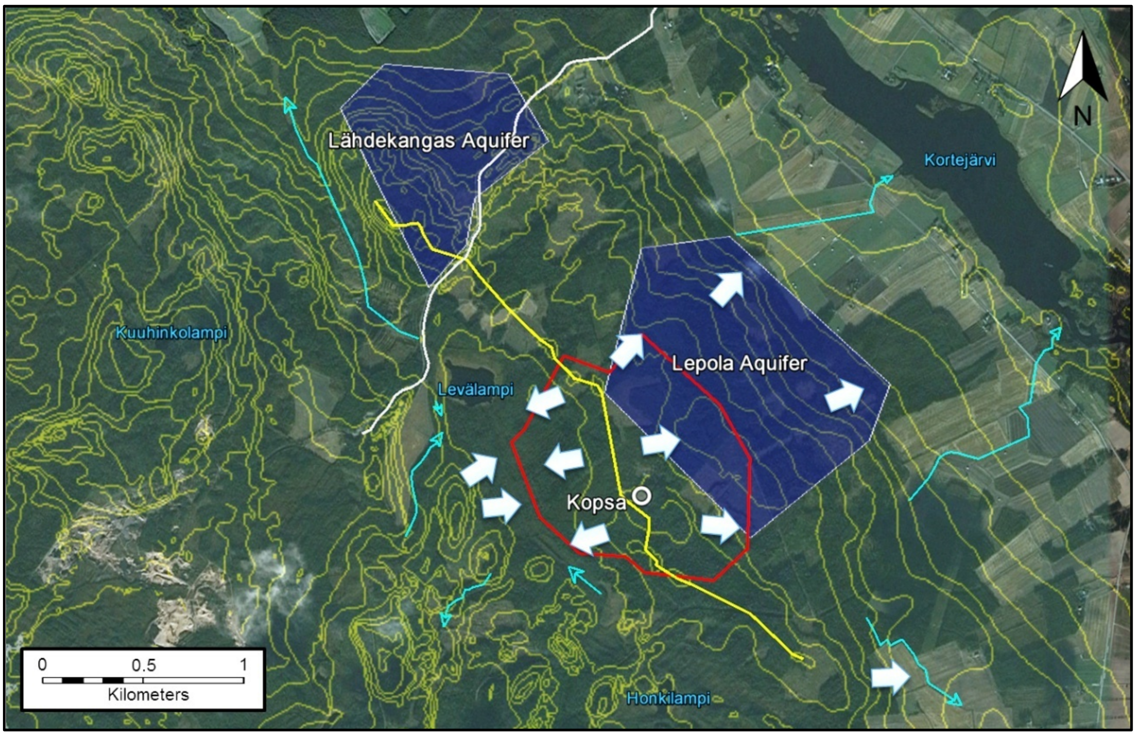


Figure 20-15: Class I (Lähdekangas) and Class III (Lepola) aquifers in the vicinity of Kopsa* (Source: Belvedere, 2013)

**Blue polygons: Approximate aquifer areas; Grey polygons: Sorting and storage facilities; Grey line: Road; Purple polygon: Mineralization area; Red polygon: Explosives storage; Red line: Applied Mining Licence Area*

The gold is associated with sulphide minerals, mainly arsenopyrite, chalcopyrite, pyrrhotite and minor pyrite. Levels of sulphur in all material types are approximately 0.7% and 0.5-0.6% and arsenic is 0.5% and 0.4% respectively.

There are a number of protected sites in the vicinity of Kopsa. Jämsänkallio is a 40 ha Natura 2000 site (No. FI1002007) located approximately 3 km south west of the mine. Both the coniferous woodland habitat at this site and certain animal species inhabiting the woodland are of conservation importance. Vihtanevan Aarnimetsä is a 12 ha Natura 2000 site (No. FI1002018) approximately 6 km southwest of Kopsa near Lake Niini. The site contains mixed woodland and bogs and animal species of conservation importance. Humalaojan and Virtain Palstan Iso Saari are smaller (less than 10 ha) private nature reserves 3.8 km SSE and 2.9 km ENE of the proposed mine. Virtain Palstan Iso Saari is an island reserve on the Kalajoki River (Figure 20-14). The reserves are protected under Chapter 3, Section 24 of the Finnish Nature Conservation Act. Springs in their natural state have the highest bio-diversity in the vicinity of the mine. Threatened plant species *Haploporus odorus* and *Leptoporus mollis* in the Jämsänkallio Nature 2000 site have been known to occur in the mine area. A grass species identified in the concession called ‘orpisorsimo’ has conservation importance under Finnish legislation.

Oksava and Haapajärvi towns have populations of 1,093 and 7,640 people respectively (Population Register Centre of Finland, 2013). Oksava and Haapajärvi are 4 km north and 4.7 km south east of Kopsa respectively. The approximate distance and direction of smaller communities near Kopsa is: Lähdekangas (2 km north); Kytöperä (4.7 km northwest); Mökkiperä (2 km south east); and Tuomiperä (7.8 km south west). The nearest households are between 0.8 km and 1 km northwest of the concession and a similar distance along two of the three un-surfaced tracks leading to the concession (Figure 20-15). Some houses may be temporarily occupied summer houses.

The mine is situated amongst privately cultivated coniferous forests. About 2 km west of the mine are gravel workings, which are accessed via a private road off road 7630 (Figure 20-15). Road 7630 runs parallel to Kalajoki River east of the mine site and is surfaced for the majority of its length.

Hitura: Hitura is located in a flat basin in the Kalajoki River valley. Figure 20-16 shows the mine and its surroundings. The river is approximately 0.5 km east of the open pit. Lake Pidis is approximately 6 km upstream of the mine. Hitura mineralised rocks comprise several nickel sulphide minerals and carbonates. Part of the TMF is underlain by porous moraine.

There are Natura 2000 sites in the region of Hitura. Pitkäneva (No. FI1002015), several kilometres south west of Hitura, has several protected birds species. Rimpinevan-Linttineva (No. FI1002014) and Rimpinevan Linnustonsuojelualue (No. FI1002014), some 10 km north west of the mine, has protected habitats and bird species respectively.

Nivala is the most populous community near Hitura. The town is 10 km north of the mine and has a population of 11,053 people (Population Register Centre of Finland, 2013). There are a number of smaller communities in the vicinity of Hitura, the nearest of which is Aittoperä and Töllinperä, with households roughly within 1.5 km the TMF respectively. Some homes are located a few hundred metres from the TMF and concentrator. Numerous homesteads occur along Road 7630, which passes between the open pit and TMF and continues southward towards Kopsa. Numerous agricultural (dairy and beef cattle) allotments extend from the eastern boundary of the TMF to the Kalajoki River. The open pit and waste rock dumps are almost completely surrounded by agricultural land. Farming is a significant source of people's livelihoods. Privately cultivated forests are located west of the TMF. Local industries include mechanical workshops, diamond drilling, trucking, bookkeeping, cleaning, catering and health services. Many of these local industries are contracted to the mine. At least one school is adjacent to Road 7630. There is a small scale historical pit approximately 1 km south west of the TMF at Makola (not shown in Figure 20-16).

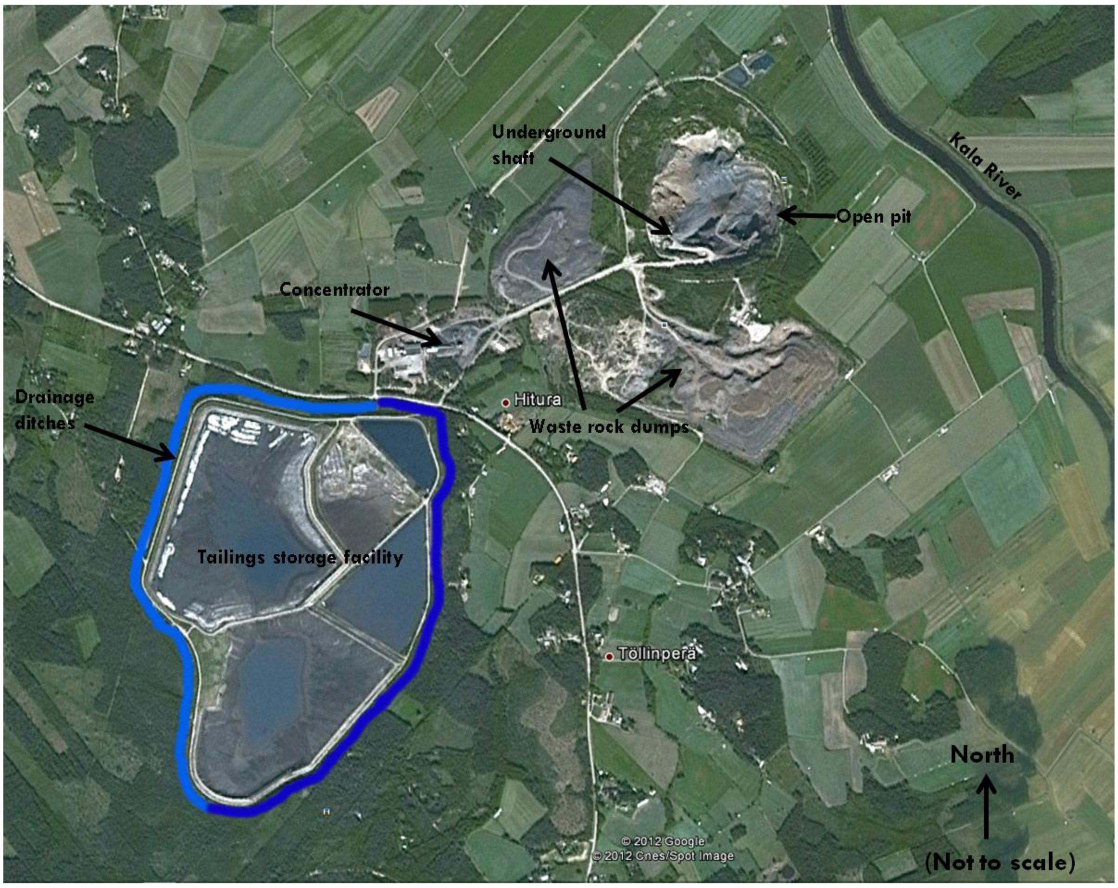


Figure 20-16: Hitura mine and surroundings (Source: Belvedere, 2013)

20.5.3 Approach to Environmental and Social Management

There is no formal environmental and social management system (ESMS) at Hitura that could be rolled out to the new mine at Kopsa. Currently the main function of the environmental department is complying with conditions in Hitura's environmental permit and obtaining environmental authorisation for the Project. SRK understands inspections of the mine's departments takes place periodically but has not seen records of these.

Belvedere subscribes to the Global Reporting Initiative's (GRI) G3.1 sustainability reporting framework and the mining and metals sector supplement to measure and report on economic, environmental, social and governance performance. Belvedere's sustainability reporting is part of the Company's strategic approach to sustainable development and environmental stewardship and demonstrates its clear commitments, risk management and performance targets. SRK has not commented on the Belvedere's GRI reporting.

Belvedere also subscribes to the International Council of Mining and Metals (ICMM) principles, which represent a voluntary best practice code of conduct set-up by industry peers to promote best practice throughout the mining industry worldwide.

Belvedere consults communities when revisions of its environmental permit are required. Although there is no formal grievance mechanism in place, people raise concerns with local authorities in Nivala, the general manager or environmental coordinator.

Concerning the Project, Belvedere has met with landowners on one occasion at the time of writing this report. (Section 20.5.5). Future meetings with stakeholders are planned as part of the on-going EIA process.

20.5.4 Environmental and Social Approvals

The status of the Project's primary environmental authorisations is discussed in this Section. Belvedere must obtain the following primary authorisations for the Project:

1. Environmental permits
2. Mining concessions
3. Water rights
4. Building rights
5. Land rights
6. Derogation permits

SRK understands Belvedere plans to simultaneously apply for a new environmental permit for Kopsa and revision of Hitura's current environmental permit to account for modifications associated with the Project. Belvedere has initiated environmental studies to produce a single EIA report and Permit Application report (discussed below) in support of both the new Kopsa permit and revised Hitura permit.

Environmental permits: In Finland, a project proponent must apply to the Regional State Administrative Agency (AVI) for an environmental permit for all activities that could pollute air, water and soil. In the case of the Project the permitting authority is the North Finland Regional State Administrative Agency / Pohjois-Suomen Aluehallintovirasto (PSAVI).

<u>Kopsa – new environmental permit</u>: Belvedere has consulted PSAVI and the authority determined that new mining components at Kopsa require a new environmental permit. The permit application process is mainly legislated by the Environmental Protection Decree (169/2000), Environmental Protection Act (86/2000), Water Act (264/1961) and Water Decree (1560/2011). The environmental permit will define conditions for the design, building, operation and closure of the new mine.

Due to the scale and nature of the planned development at Kopsa, an EIA is required for the new mine and Belvedere must submit an EIA report in support of the new environmental permit application. The EIA report approval process is administered by another authority, namely Einkeino, Liikenne- ja Ympäristöministeriö / The Centre for Economic Development, Transport and the Environment (ELY) (under the Ministry of Environment). The EIA must be carried out according to procedures set out in EIA Law 468/1994 and Act 713/2006. During the first phase in the EIA reporting process, an EIA Programme is developed and submitted to ELY. The programme must present a conceptual description of the project, its alternatives, salient environmental and social aspects, EIA methodology and authorisation process going forward. The EIA report must be prepared on the basis of comments from authorities and the public on the EIA Programme. In addition to the EIA report Belvedere must also compile a Permit Application report. After submission to ELY, the authority will issue a statement on the EIA report, which must be appended to the Permit Application report together with the EIA report. The Permit Application report is submitted to PSAVI in support of the environmental permit application.

Figure 20-17 shows the environmental permit application process following submission of the Permit Application report to PSAVI. The public will have an opportunity to comment on the application. Belvedere will respond to comments and PSAVI will base its record of decision (RoD) on the environmental permit on all available permitting documentation, comments and responses. If AVI's RoD is appealed, then Belvedere may begin constructing the Project before the appeal procedure (which could take a number of years) is finalised providing that Belvedere has obtained the requisite permits (see below) to start preparatory works and operations and placed appropriate guarantee as stipulated in the RoD.

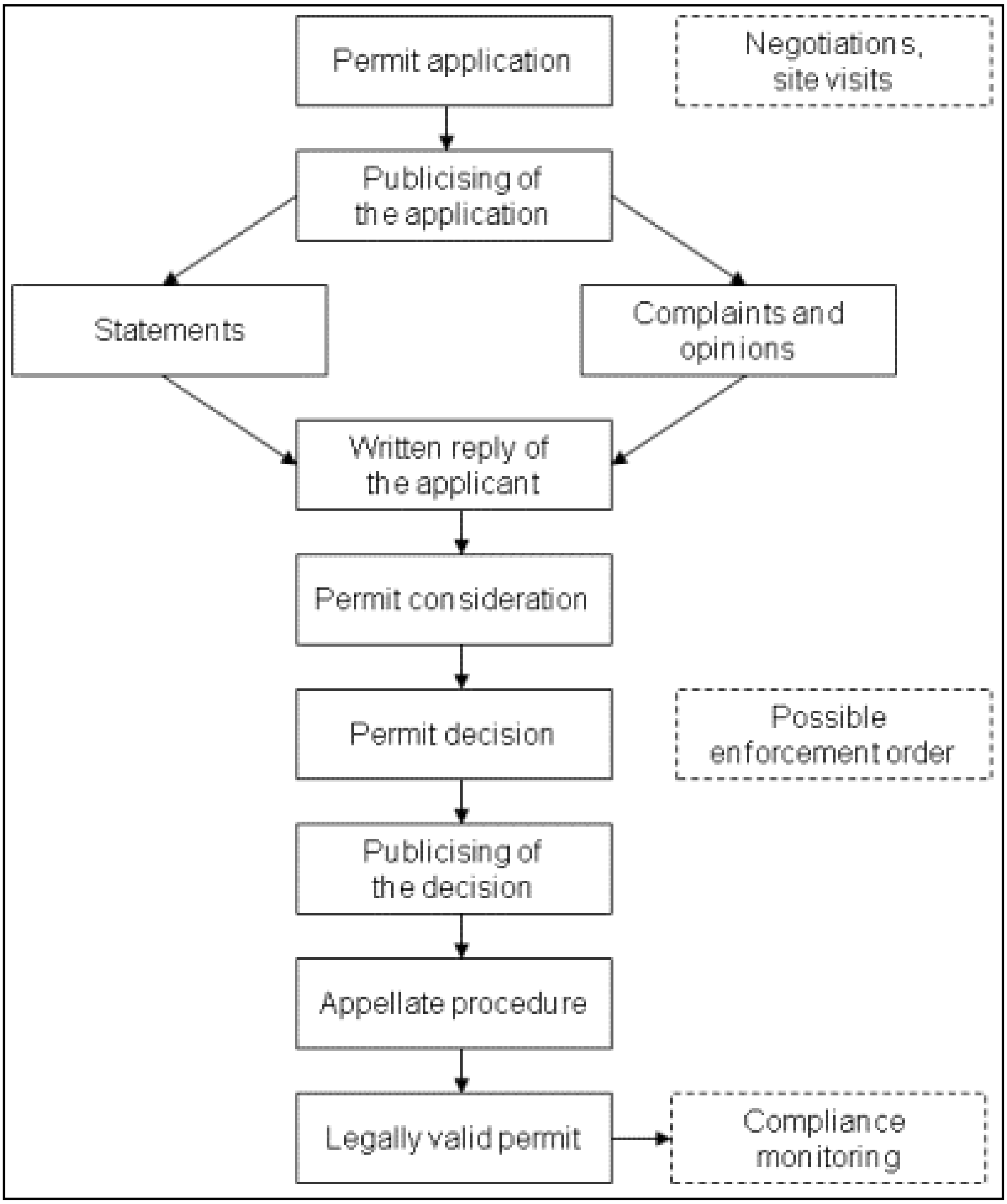


Figure 20-17: Environmental permit application process (Source: Ministry of Environment, 2013)

Hitura – revised environmental permit: PSAVI granted Hitura its current environmental permit (No. 66/10/1) 13 August 2010 to re-start operations. Belvedere will apply for the following two revisions of this permit associated with the Project:

- *Revision 'A'* for modifications at Hitura associated with the Project; and
- *Revision 'B'* to increase the height of the existing TMF perimeter wall.

The requirement for EIA for revising an existing environmental permit is determined by ELY on a case by case basis and depends on the nature and scale of the development. If required, the process outlined in Figure 20-17 must be followed following submission of the Permit Application report.

Revision 'A': Belvedere has consulted PSAVI and the authority determined that EIA is required in support of the revision to Permit No. 66/10/1 to account for the following modifications at Hitura:

- Deposit Kopsa gold-copper tailings in one of the two new cells at the TMF. Belvedere is authorised in its current environmental permit to construct and operate the two new 65 Ha and 25 Ha cells for nickel tailings;
- Mine 2.2 Mtpa total material and produce no greater than 650 Ktpa tailings; and
- Modifications to the existing nickel concentrator to convert it to process gold-copper material from the new mine at Kopsa and a new sulphide concentrate storage facility.

Revision 'B': Belvedere requires this revision to Permit No. 66/10/1 to increase the height of the existing TMF perimeter wall to 111.5 m. Belvedere's preferred choice is to deposit Kopsa copper-gold tailings onto the expanded TMF. Belvedere believes Permit No. 66/10/1 may be revised for this Project component without EIA if copper-gold tailings are demonstrated to be as or more benign than existing nickel tailings.

20.5.4.1 Status of the Project's environmental permitting:

Belvedere's estimate of critical path environmental authorisation milestones is given in Table 20-22.

Table 20-22: Company estimated critical path for Project environmental authorisation milestones

	Best case	Worst case
EIA REPORT		
Issue EIA Programme	Oct. 2013	Nov. 2013
Issue EIA Report	Q2 2014	Q4 2014
Statements on EIA Report & finalise report	Q4 2014	Q1 2015
ENVIRONMENTAL PERMIT REPORT		
Issue Permit Application report	Q3 2014	Q4 2014
Final record of decision	Q3 2015	Q1 2016

SRK has the following observations on the status of the Project's environmental permitting:

- Belvedere has started collecting baseline water quality monitoring data at Kopsa. Certain baseline ecology studies have been completed (vegetation, fish and benthic organisms).

- A formal meeting with affected landowners at Kopsa took place 18 April 2013.
- Belvedere plans to submit the EIA programme to authorities during the fourth quarter of 2013.
- Belvedere plans to submit the EIA report to ELY in either the second quarter of 2014 (best case) or the fourth quarter of 2014 (worst case).
- ELY could issue a statement in either the last quarter of 2014 (best case) or first quarter of 2015 (worst case).
- The Permit Application report could be submitted to PSAVI during the third quarter of 2014 (best case) or fourth quarter of 2014 (worst case).
- The current estimated timeframe for PSAVI to issue a statement on the Permit Application report is one year from the announcement date, i.e. last quarter 2015 or first quarter of 2016.
- PSAVI could grant the environmental permit for Kopsa and revised permit for Hitura during the third quarter of 2015 (best case) or first quarter of 2016.

Mining concessions: Belvedere has applied to the Finnish Safety and Chemicals Agency (Tukes) for a mining concession for Kopsa and anticipates the concession will be granted mid 2014. At the time of writing Belvedere was responding to comments from Tukes. An application for a mining concession takes place in two phases. In the first phase Tukes issues a RoD on the application and thereafter (Phase 2) instructs the local land survey authority to officially survey the concession. A final mining concession is granted after both phases are completed. Because Belvedere applied for the concession in 2008 under the old Mining Act (503/1965), no EIA report was required in support of the Kopsa mining concession application. Notwithstanding this, Belvedere must obtain aforementioned environmental permit and the water, building and land rights discussed below.

Concerning Hitura, Tukes has granted Belvedere an extension of Hitura's mining concession to accommodate the new TMF cells.

The new Mining Act 621/2011, which came into force 1 July 2011 requires the EIA report and statement to be appended to future mining concession applications, assuming the development in question requires EIA.

Water rights: Applications for a water permit are made to PSAVI in terms of the (587/2011) and Water Decree (1560/2011) together with the Permit Application report. PSAVI issues the water permit in conjunction with the environmental permit. Belvedere will require a water permit for the following Project components, which SRK understands are being addressed in the EIA:

- Dewatering of the open pit and water discharge either to the Kalajoki River or Lake Haagan;
- Discharge to the Kalajoki River from the Hitura TMF (Section 20.1); and
- Stream crossings along transport routes.

Building rights: Belvedere will have to compile local (and possibly regional) building plans (in terms of the Building Act 132/1999) and land use amendments for approved by local municipalities. The EIA statement is required before the plans and amendments can be submitted and approved. Following approval, building permits are required from the municipalities prior to commencing construction.

Land rights: Belvedere's building permit application must demonstrate right of access to land in the mining concession. Thus far, Belvedere has elected to purchase land in the Kopsa mining concession from landowners. It has not been determined how Belvedere plans to access land in the extended concession at Hitura.

Derogation permits: Belvedere may require derogation permits in terms of the Water Act (587/2011) if mining activities impact on natural springs. Derogation permits may also be required in terms of the Nature Conservation Act (1096/1996, 48 and 49) and Forest Act (1093/1996, 10 and 11) if activities impact on certain plant species, habitats and forests of conservation importance at the mine site. The permits, which would be issued by ELY, would be required before Belvedere applies for an environmental and building permit.

Load limits on truck loads: SRK understands The Ministry of Traffic and Communications (Liikenne ja Viestintäministeriö) is planning a new decree that would permit an increase in the maximum loads of trucks on Finnish roads. Loads would increase from the current 60 t limit (including the weight of the vehicle and an additional 40 t load total weight) to 76 t. This is roughly a 10 t increase in load carrying capacity with larger trucks. It is understood the ministry's goal is to validate the decree within one year.

<u>Observations on environmental and social approvals:</u> SRK has the following observations on environmental and social approvals:

- As of end September 2013 the draft EIA programme had not been submitted to the authorities.
- There is a high level of uncertainty on the timeframe to complete the Permit Application report to meet PSAVI's requirements for the environmental permit. In most cases it takes one year before the authority announces the application. Current worst case projections for issuing the environmental permit could be further extended to the first quarter 2017. SRK understand authorities are taking steps to reduce permit review periods but it is not known whether this will benefit the Project.
- There was a non-conformance with permit conditions with respect to pH (values less than 6) in January 2013. Belvedere informed the authority but received no response.
- In SRK's opinion no water permit is needed for optical sorting because the activity will not require washing of the material and no effluent will be generated.
- Options to purchase or lease land in the footprint of the extended mining concession at Hitura (for the new TMF cells) and new mining concession at Kopsa have yet to be finalised.
- Obtaining derogation permits would normally not present a major risk to the environmental approval process.
- Changes to the regulatory framework could transpire from authorities' review of mines in Finland - see 'stress test' issue below.

20.5.5 Environmental and Social Issues

Based on the review undertaken by SRK, the principal environmental and social issues and or liabilities relating to the asset/s are listed below.

Potential delay in the Project's environmental authorisation: Refer to SRK's observations on environmental and social approvals (Section 20.5.4).

Interests opposing the Project: Belvedere has identified the following interests opposing the Project:

- Some landowners in the Kopsa mining concession are demanding unrealistic prices. If expropriation is required this could protract the land acquisition process.
- Users of the private road accessing gravel workings (Alternative 1 - Figure 20-14, preferred option) - early indications of resistance to the mine using of this road to transport material to Hitura. Another road access alternative route could result in higher capex and opex expenditure.
- Users of natural springs in the vicinity of Kopsa could oppose the Project because they perceive the mine will further decrease the quality of water.

Water management: Dewatering the open pit at Kopsa will lower groundwater levels in the Lepola aquifer, part of which is located within the footprint of the pit. Whilst the aquifer is not used for domestic water (due to its poor quality), the impact of lowering groundwater levels beneath abundant agricultural land and on springs of conservation value will need to be assessed for environmental permitting.

Impacts on the Class I Lahdekangas aquifer, which supplies domestic water to a number of households, will have to assessed.

As part of the EIA, Belvedere is determining the potential for waste rock to generate acid and leach metals (acid rock drainage and metals leaching – ARDML) such as arsenic (Section 20.2). Preliminary results indicate arsenic may be mobalised. Acid generation tests are not yet conclusive. Surface water runoff and leachate from the waste rock dump may have to be contained with a liner and treated before discharge to Kalajoki River or Lake Haagan. Pit water may also have to be treated before discharge. As suggested in the geochemistry Section 20.2, costs for these water management facilities have been assumed in the PEA economic model.

At Hitura, whichever scenario Belvedere selects for disposal of copper-gold tailings, PSAVI could place stricter limitations (notably on cyanide, arsenic and copper) in the revised environmental permit on the quality of process water discharged to the Kalajoki River. Limitations in the current permit are: a) nickel concentrations must be less than 2.5 mg/l or less than 200 kg/year; b) pH - must be between 6 and 8.5; c) particulate matter - may not exceed 20 mg/l; d) flooding of the natural drainage ditch is not permitted; and e) nitrogen and phosphorus (nutrients essential for plant growth) must be monitored. Cyanide will be removed from the water in a cyanide destruction plant; however seepage of arsenic in copper-gold tailings to groundwater beneath the existing unlined TMF may be an issue if tailings are deposited on the existing dam. Moreover the arsenic content of water discharged from the TMF may be an issue.

In 1998, porous glacial moraine beneath part of the TMF transmitted process water with elevated nickel and sulphates to nearby aquifers and contaminated the drinking water of household's south east of the TMF. Consequently the Töllinperä water cooperative was supplied with municipal water, which Hitura financed up until the first quarter of 2013. In 1999/2000 clay barriers were constructed and today dewatering wells pump groundwater back to the TMF. It is understood this issue has largely been resolved. If copper-gold tailings are deposited onto the existing TMF, then treatment may be required prior to deposition to prevent seepage of toxic levels of arsenic from further contaminating groundwater. As suggested in the geochemistry Section 20.2, costs for a water treatment facility at Hitura have been assumed in the PEA economic model.

The Ministry of Environment, Labour and Ministry of Agriculture and Forestry began 'stress testing' mines in June 2013 to assess whether the mines have sufficient water storage, treatment and diversion capacity to cope with exceptionally high rainfall events. Belvedere has been requested by the authorities to complete the assessment. The outcome of these tests is not known at this point.

It is understood from Ahma (Belvedere's local consultants carrying out EIA), that environmental studies will include assessment of cumulative impacts prior to submission to the authorities. Cumulative impacts of Hitura and Kopsa discharging water to the Kalajoki River could lead to authorities imposing stricter discharge limitations.

Disturbance to communities along part of Road 7630 towards Hitura by trucks: The full magnitude of impacts of this activity on noise, air quality and health and safety and the views of communities should be fully understood in the EIA for Kopsa. A preliminary estimate of the frequency of trucks (one passing through community approximately every five minutes (based on restriction to a 40 t truck operating during the daytime to achieve a production rate of 1 Mtpa), may be opposed by these communities and an alternative route may be required resulting in higher OPEX costs. The optical sorting option, should this be used, will reduce the frequency of truck movements through the community. The views of communities will be solicited in stakeholder consultation carried out on behalf of the EIA.

At least one property from the mine to Road 7630 (Alternative 2, Figure 20-14) will have to be purchased.

Noise and dust complaints from communities: Dust emissions from the Hitura crushing station were remediated with the installation of a new dust control device in 2011. This device also reduced noise levels. According to Belvedere, some dust still continues to spread from TMF during windy storms at spring time.

20.5.6 Closure Requirements and Costs

Belvedere has compiled and submitted to the authorities a closure plan for Hitura dated December 2012 as a requirement of the environmental permit. Belvedere is waiting for the plan to be approved. According to the Company all closure and rehabilitation costs have been provided for in a bank account via a bank deposit, which may only be redeemed with authorisation from ELY.

Belvedere has consulted authorities about the temporary suspension of works at Hitura. SRK understands that Belvedere does not have to implement closure measures in the plan at this time. A revised programme for closing the waste facilities may be required in the authorities' statement on Belvedere's updated 2013 closure plan; a decision is expected in December 2013. The only closure requirement in the current environmental permit is to start closing the existing TMF 1 year after the new TMF cells are constructed. In terms of future closure requirements, a 'safety deposit' will be required in 2014. This deposit will be monitored by Tukes and will cover safety aspects. Belvedere estimates the cost will be €100,000 of which €50,000 is provided for safeguarding the open pit.

Concerning closure of Kopsa, there is a requirement in the current environmental permit for Belvedere to provide a bank guarantee or deposit for the following amounts (totalling **€**1,500,000) to close the new cells at the Hitura TMF:

- €500,000 before starting constructing the new 65 Ha tailings cell;
- An additional €500,000 must be provided when the 65 Ha tailings cell becomes operational; and
- An additional €500,000 before starting constructing the new 25 Ha tailings cell.

Closure plans and costs for Kopsa incorporating the two new tailings cells at Hitura, mine waste facilities at the Kopsa mine site and water treatment at Hitura and Kopsa are provided elsewhere in this report.

20.5.7 Overview of Findings

The Project has a complex and lengthy approval process ahead with some uncertainty on when the environmental permit will be issued. Early robust impact evaluation is critical to reduce the risk of authorities and the public discrediting the study and delaying authorisation.

The main environmental issues relate to water management. Project mineralization contain sulphides with elevated levels of arsenic. There is an excess of water and it will be necessary to continue discharging water from the open pit and process. Authorities could impose stricter limitations on the quality of water discharged to protect receiving environments. Water containment and treatment facilities maybe required and these have been accounted for in both operating and capital costs assumptions in the PEA economic model.

Other receptors include communities along part of the transport route and some road users and land owners. Less economically favourable transport routes may have to be considered to mitigate disturbance and risks to local communities. These various alternatives will be investigated further as part of the next phase of study.

20.5.8 Risks

A summary of Project environmental and social risks follows:

- Delay in obtaining environmental authorisation;
- Higher CAPEX and OPEX due to potential requirement for water containment and treatment facilities;
- Higher CAPEX and OPEX due to potentially longer transport route from Kopsa to Hitura; and

- Increased community opposition to the Project if transport and land use issues are not resolved.

20.5.9 Recommendations

SRK has the following recommendations for the feasibility study:

- Detailed synchronisation of the schedules for the various Project and environmental work packages to confirm all outputs are aligned.
- As soon as practicably possibly, carry out a detailed review all anticipated Project environmental and social aspects, potential impacts and specialist studies. This would minimise the risk of missing / omitting critical aspects for EIA and delaying the project.

Separate to Project environmental authorisation, develop a formal ESMS. The ISO14001 international standard for ESMSs system is commonly implemented at mines (and other industries) worldwide. Belvedere does not have to seek ISO14001 accreditation but could benefit from some of the system's systematic approach to impact assessment and management. The system also provides a framework within which the mine captures and monitors compliance with its legal, policy and other requirements. Effective ESMS (or certification to ISO14001) should give greater confidence to authorities and investors Belvedere is effectively managing its environmental and socials issues and commitments and is well positioned to incorporate potential stricter environmental requirements.

21 CAPITAL AND OPERATING COSTS

21.1 Introduction

The following sections provide an overview of operating and capital cost assumptions for the Kopsa PEA. These costs are described in more detail in previous sections of this report. In most cases, costs were estimated by SRK, notable exceptions being the operating costs for the Hitura process plant and tailings facility, which have been based to a large extent on actual operating costs from 2012.

Whilst the designed pits have been scheduled at four different production rates (500 Ktpa, 750 Ktpa, 1.0 Mtpa and 1.2 Mtpa), considering two different processing scenarios, as presented in Table 21-1 below, only capital and operating costs for Scenario 6 (production rate of 1.2 Mtpa with sorting) are discussed in this section.

Table 21-1: Production rates and processing scenarios considered as part of this PEA

Scenario	Production Rate (Mtpa)	Sorting
1	0.5	Without sorting
2	0.75	Without sorting
3	1.0	Without sorting
4	1.0	Sorting
5	1.2	Without sorting
6 = (base case)	1.2	Sorting

21.2 Operating Costs

An overview of operating costs for the major costs centres is presented in Table 21-2 and illustrated in Figure 21-1 over the Project life of mine.

Table 21-2: Overview of operating costs by major cost centre

	USD/t moved	USD/t milled	Percentage of total
Mining	5.9	27.4	53%
Processing	3.4	15.9	30%
Tailings	0.6	2.8	5%
Environmental & Closure	0.2	1.1	2%
G&A*	0.5	2.3	4%
Contingency	0.5	2.5	5%
Total	11.1	52.0	100%

**G&A based on 2012 actual costs.*

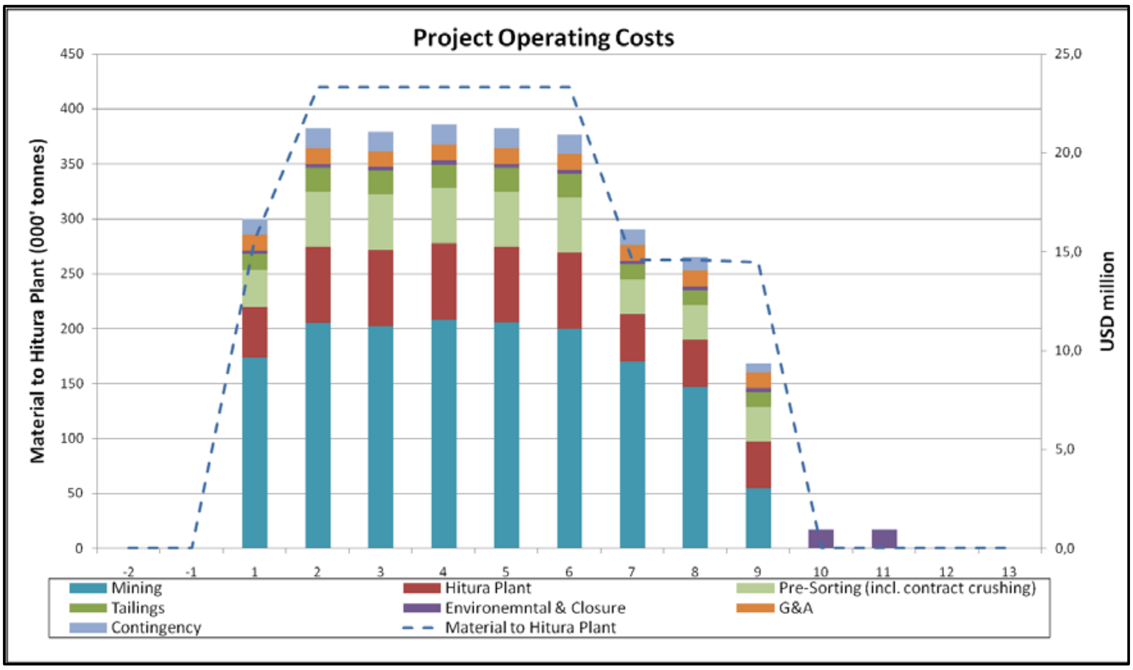


Figure 21-1: Summary of operating costs over the life of mine (Source:SRK, 2013)

21.2.1 Mining

Assumed operating costs are presented below. These estimated costs are based on the selected mining production schedule (1.2 Mtpa) and corresponding equipment usage. Increasing costs with pit depth are accounted for as is the cost of re-handling material from stockpiles into haul trucks.

Table 21-3: Mine operating costs

Mining Cost Centre	USD / tonne total material
Drilling	0.04
Blasting	0.24
Loading	0.25
Hauling_In pit	0.34
Stockpile Excavation	0.14
Haulage_Mine to plant	0.49
Mobile Mining Equipment	0.39
Auxiliary Equipment	0.20
Labour	3.49
Mine Facilities & Other (incl. grade control)	0.30
Total Mining	5.88

21.2.2 Processing

Table 21-4 presents the assumed operating costs for processing Kopsa material. These costs reflect Scenario 6 (Table 21-2), with a process throughput at the Hitura plant of 0.35 Mtpa.

With the exception of sorting, flotation and cyanidation, these costs are based on 2012 actual costs at the Hitura plant from processing of nickel sulphide ore from the Hitura underground mine.

Table 21-4: Process operating costs

Processing Cost Centre	USD / tonne
Kopsa on-site sorting	
Sorting including on-site crushing	2.33
Hitura process plant	
Grinding	2.53
Flotation	4.13
Cyanidation	1.00
Filtering	0.67
Process general	0.43
Repair shop	0.43
Laboratory	0.00
Total Processing	9.19

21.2.3 Tailings

As presented in Table 21-2 above, assumed operating costs for disposal of tailings is USD 2.8 per tonne of material processed at Hitura, based largely on 2012 actual costs received from the Company. This cost also includes an estimate of EUR 67 for every tonne of high sulphide tailings, which for the purposes of this study are assumed to be bound with cement to form a paste backfill and pumped to underground workings at Hitura.

21.2.4 Environmental, Rehabilitation & Closure

As presented in Table 21-2 above, operating costs in this area amount to USD 1.1 per tonne of material processed, or M USD 3.7 over the life of mine. The majority of these costs (M USD 1.9 million) are running and maintaining the water treatment facilities at the Hitura TMF and Kopsa waste rock dump.

The total provision for closure of the Kopsa waste rock dump and the new cells at the Hitura TMF is M USD 1.8.

21.2.5 Treatment Charges and Refining Costs

In addition to the costs presented in Table 21-2 above, the following treatment charges and refining costs (TCRC's) have been are assumed.

Table 21-5: Treatment Charges and Refining Costs

TCRC's	(Unit)	Cost
Cu Treatment Charge	**(USD/t)**	`63
Cu Refining Charge	**(USD/lb)**	0.063
Au Refining Charge	**(USD/oz)**	5.0
Ag Refining Charge	**(USD/oz)**	0.5

21.3 Capital Costs

The capital costs estimated as part of this study have been derived mostly by SRK and are discussed in detail elsewhere in this report. The following section presents a summary of these costs, which total M USD 48. SRK notes the following:

- Contingencies of 25% have been applied to all capital costs;
- Working capital has been assumed at 20% of first production year operating costs;
- No provision has been made for sustaining capital, which for the purposes of this study is accounted for in operating cost provisions.
- In general (with the exception of tailings construction), capital costs have been profiled with 70% of expenditure occurring in the first pre-production year and the remaining 30% occurring in the first year of production.

Figure 21-2 gives a breakdown of the envisaged capital expenditure over the life of mine and split between the major cost centres, including contingency and working capital.

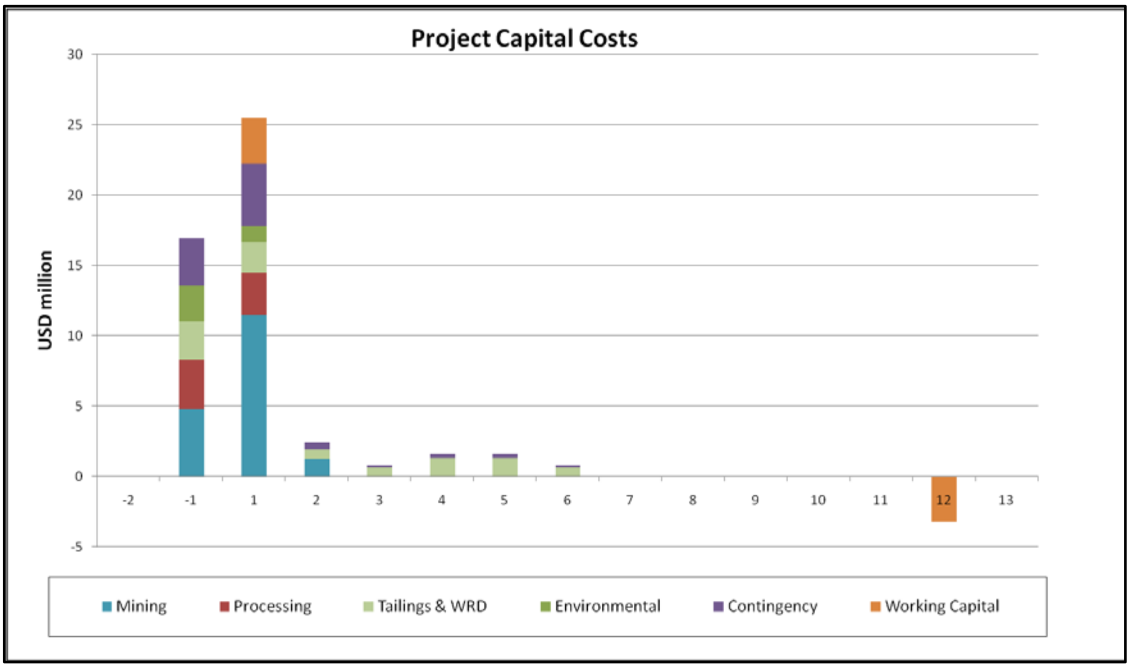


Figure 21-2: Capital cost breakdown over the LOM (Source:SRK, 2013)

Table 21-6 below presents capital cost assumptions, with a high-level breakdown under the major costs centres. Roughly 90% of capital is assumed to be required in the first pre-production year and subsequently the first two years of production.

Table 21-6: Capital cost assumptions

Description	Value (USD million)
Mining	
Mine Facilities & Haulage Dispatch System	6.1
Haul Roads	0.7
Mobile Mining Equipment	9.0
Auxiliary Equipment	2.1
Total Mining	**17.9**
Processing	
Sorting units & construction	2.2
CIL plant & refurbishments to Hitura mills	5.0
Total Processing	**7.2**
Tailings & WRD	
SRK estimate tailings construction costs	13.1
Reduction through EU Life Project funding	-6.6
Tailings back-fill plant for high sulphide material	0.3
WRD Construction (incl. ground prep & liner)	2.6
Total Tailings & WRD	**9.5**
Environmental	
Water Management Facilities (Hitura & Kopsa)	1.0
Water Treatment Plants (Hitura & Kopsa)	2.7
Land purchase (Kopsa & Hitura)	0.4
Total Environment	**4.1**
Contingency (25%)	9.7
Total	**48.3**

22 ECONOMIC ANALYSIS

SRK has constructed a technical economic model (TEM) to derive a post-tax Net Present Value (NPV) for the Kopsa Project. The TEM is based on the technical assumptions developed by both the Company and from work undertaken by SRK, as commented on in the previous sections of this report. The Company has provided SRK with the processing physical parameters, refining/smelting charges and various assumptions for operating and capital costs, which in some cases are based on 2012 actual costs incurred at Hitura during processing of nickel sulphide ore. SRK has reviewed these assumptions and has adjusted these where appropriate to reflect the views as presented in previous sections of this report.

The economic analysis contained in this report whilst including Measured and Indicated Resources only, is still preliminary in nature. Conversion of these Measured and Indicated Mineral Resources to Mineral Reserves would require the support of a pre-feasibility level study. There is no certainty that the reserves development, production, and economic forecasts on which this Preliminary Assessment is based will be realised.

22.1 Valuation Process

22.1.1 General Assumptions

The model is based on production from a single open pit mine at the Kopsa site, with on-site crushing and possible sorting based on X-ray transmission (XRT) technology. The model assumes material is trucked to the Company's existing processing facility at Hitura for production of a marketable copper sulphide concentrate and smelted gold/silver doré through conventional flotation, cyanide leaching and Carbon-in-Pulp (CIP) / Carbon-in-Leach (CIL).

The designed pits have been scheduled at four different production rates (500ktpa, 750ktpa, 1.0Mtpa and 1.2Mtpa), considering sorting and no sorting options, as presented in Table 21-1 above. For the purposes of this report, only an economic analysis of Scenario 6 base case (ROM production rate of 1.2Mtpa with sorting) is discussed in detail. For illustrative purposes, a project valuation for each scenario is presented in Table 22-6 below.

As part of the NI 43-101 process, SRK has constructed a post-tax and pre-finance TEM and assumes:

- a US Dollar (USD) valuation currency, with any Euro (EUR) derived costs being converted at a EUR:USD exchange rate of 1:0.75;
- a base case discount rate of 8%;
- the TEM is in real 20113 terms and no nominal model is presented;
- due to the uncertainty of when this project may be brought into production, the start of mining is assumed to be from 'Year 1' with two pre-production years ('Year -1' and 'Year -2') for the set up of basic mine infrastructure and access;
- discounting of cashflows starts in year -1;
- working capital based on 25% of the operating costs from the first year of production;
- depreciation on a 20% declining balance basis; and
- corporate tax rate of 24.5%.

The TEM considers the revenue and cost implications of both a marketable copper sulphide concentrate and smelted gold/silver doré.

22.1.2 Commodity Price Assumptions

The following commodity price assumptions have been used:

Copper USD 6,000 / tonne

Gold USD 1,200 / troy ounce

Silver USD 20 / troy ounce

22.2 Mine and Process Physical Assumptions

22.2.1 Mining

A summary of the combined mass movement of material is presented in Figure 22-1 and below. A discussion of material movement by pit is presented in Section 16.

It is assumed that marginal material is stockpiled in the waste dump area and processed at the end of the mine life, during years 8 and 9.

Table 22-1: Summary of movement of material from the Kopsa open pit

Mining	Unit	Value
ROM	(tonnes '000)	7 565
Marginal Material	(tonnes '000)	1 479
Waste Rock	(tonnes '000)	4 175
Glacial Ovb	(tonnes '000)	1 567
Total Material Mined	(tonnes '000)	14 787
Strip ratio	(w:o)	0.63
Life of mine	(years)	9
Grade Cu	(%)	0.15%
Grade Au	(g/t)	0.91
Grade Ag	(g/t)	2.21

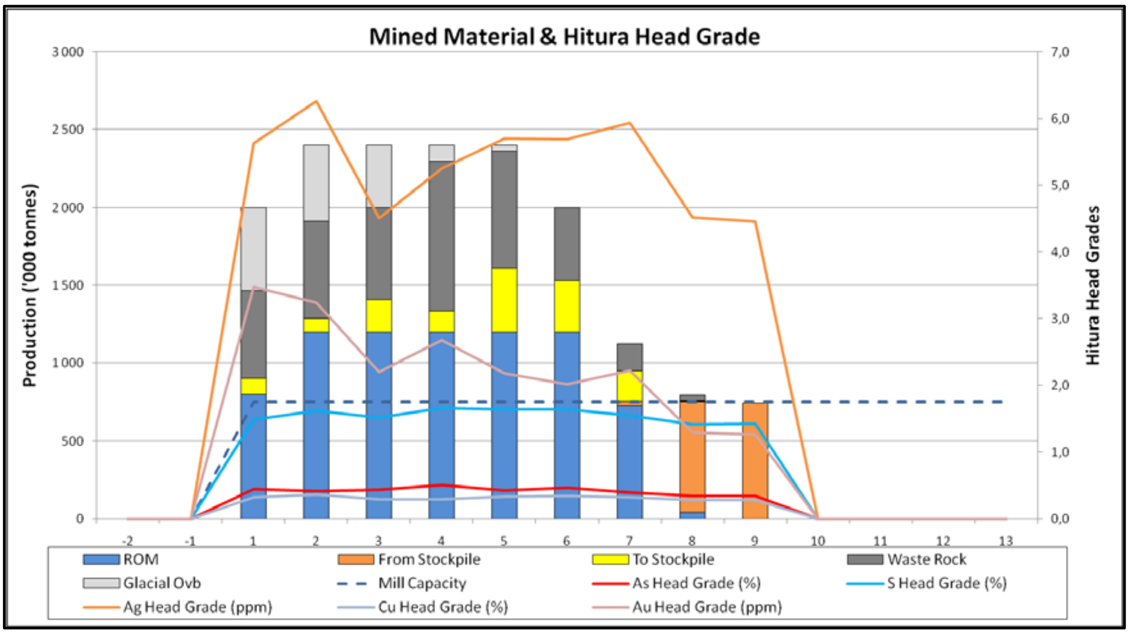


Figure 22-1: Summary of mass movement of material from the Kopsa pit (Source:SRK, 2013)

22.2.2 Process, Smelting and Refining

Process recovery and concentrate grade assumptions are discussed and presented Section 17.2, Table 17-1 above. Table 17-1 is reproduced below for the base case only (Table 22-2).

Smelting and Refining assumptions are presented in Table 22-3.

Table 22-2: Base case recovery and concentrate grade assumptions

Item		Unit	Base Case, Scenario 6 (ROM production rate of 1.2Mtpa with sorting)
RoM Production		tpa	1 200 000
Delivery to Plant		tpa	420 000
Sorting Loss	Cu	%	25
	Au	%	10
Flotation Feed Grade	Cu	%	0.32
	Au	g/t	2.34
Copper Concentrate		tpa	4 800
	Cu Rec	%	80
	Au Rec	%	40
	Cu	%	22.5
	Au	g/t	82
Sulphide Concentrate		tpa	12 600
	Au Rec	%	44.75
	Au	g/t	35.0
Cyanidation Recovery	Au	%	95
Recovery to Doré	Au	%	42.5
Overall Recovery	Cu	%	60
	Au	%	76.30

Table 22-3: Smelting and Refining assumptions

Item	Unit	Value
Copper Concentrate Losses & Deductions		
Cu Payable	(%)	95.0
Cu unit deduction	(%)	1.0
Au unit deduction	(g/t)	1
Ag unit deduction	(g/t)	30
Leach Doré		
Au Payable	(%)	99.5
Ag Payable	(%)	98.0

SRK notes that (a) no penalties have been assumed for contained arsenic and (b) there is currently no provision for the transport of copper concentrate product to the refiner. For the purposes of this study, it is assumed that these costs are covered by the deduction, treatment and refining charges outlined above.

22.3 Revenue & Cash Flow Projections

Figure 22-2 below provides an overview of net revenue by product over the life of mine.

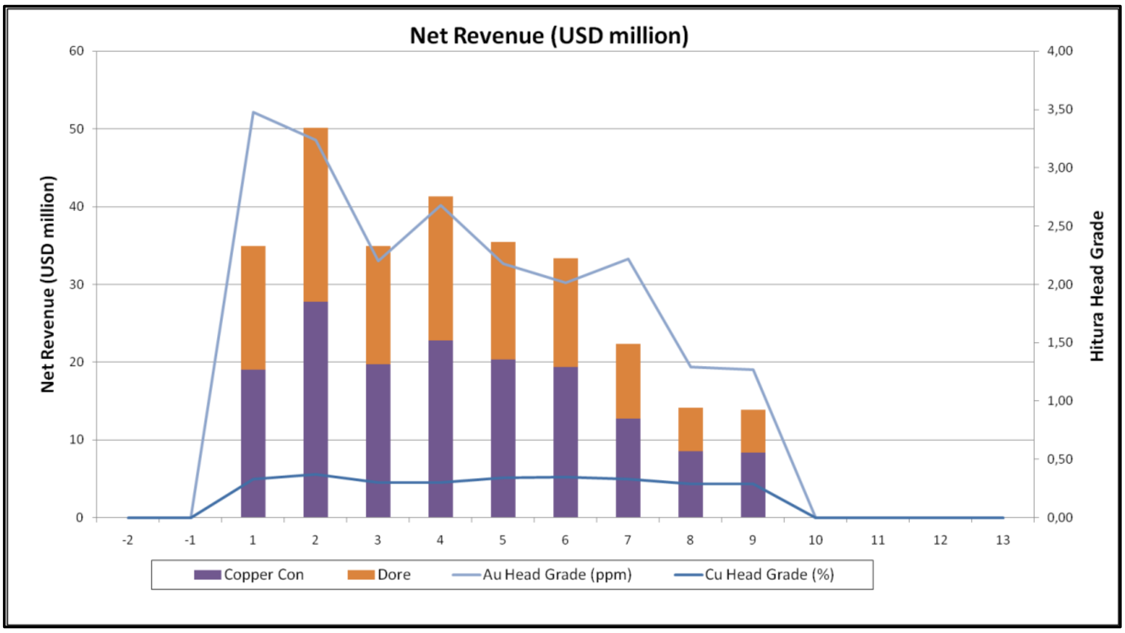


Figure 22-2: Contribution to net revenue of copper concentrate and Au-Ag doré (net of TCRC's, losses and deductions). (Source:SRK, 2013)

A valuation of the Project has been derived based on the application of Discounted Cash Flow (DCF) techniques to the pre-tax, pre-finance cash flow based on the inputs and assumptions presented in this and previous sections of this report. All figures are presented in real terms.

In summary, for the base case (Scenario 6), at a Cu price of USD 6 000/tonne and Au price of USD 1,200 / troy ounce, a 8% discount rate the project has a post-tax, pre-finance NPV of USD 25.3 million for production of both a copper concentrate and Au-Ag doré. A summary of the results of the cash flow modelling and valuation are presented in Table 22-5 and Table 22-6. A summary annual cash flow is presented in Table 22-4.

Table 22-4: Summary Annual Cash Flow

SE443-U5522 Kopsa PEA Model															
Summary Annual Cashflow	**Units**	**Total**	-2	-1	1	2	3	4	5	6	7	8	9	10	11
ROM 1.2 Mtpa, Pre-sorting															
CASHFLOW															
Mining															
ROM	**(000' tonnes)**	**7 565**	0	0	800	1 200	1 200	1 200	1 200	1 200	725	40	0	0	0
To Stockpile	**(000' tonnes)**	**1 479**	0	0	101	87	208	135	409	328	200	9	0	0	0
Waste Rock	**(000' tonnes)**	**4 175**	0	0	562	624	592	961	753	472	174	38	0	0	0
Glacial Ovb	**(000' tonnes)**	**1 567**	0	0	537	489	400	104	38	0	0	0	0	0	0
From Stockpile	**(000' tonnes)**	**1 479**	0	0	0	0	0	0	0	0	25	710	744	0	0
Total Material Mined	**(000' tonnes)**	**14 787**	0	0	2 000	2 400	2 400	2 400	2 400	2 000	1 100	87	0	0	0
Stripping Ratio (waste / ROM)	**(w:o)**	**0,63**	0,00	0,00	1,22	0,86	0,70	0,80	0,49	0,31	0,19	0,78	0,00	0,00	0,00
Processing															
Material to Hitura Plant	**(000' tonnes)**	**3 166**	0	0	280	420	420	420	420	420	263	263	261	0	0
Au Head Grade (ppm)	**(grams)**	**2,34**	0,00	0,00	3,48	3,24	2,20	2,68	2,18	2,02	2,22	1,29	1,27	0,00	0,00
Cu Head Grade (%)	**(tonnes)**	**0,32**	0,00	0,00	0,33	0,37	0,30	0,30	0,34	0,35	0,33	0,29	0,29	0,00	0,00
Copper Concentrate Product	**(tonnes)**	**36 445**	0	0	3 265	5 526	4 467	4 484	5 087	5 170	3 054	2 711	2 683	0	0
Dore - Au	**(oz)**	**100 084**	0	0	13 170	18 415	12 524	15 209	12 388	11 447	7 879	4 582	4 470	0	0
Dore - Ag	**(oz)**	**86 974**	0	0	8 071	13 459	9 684	11 289	12 259	12 223	7 975	6 065	5 948	0	0
Revenue															
Gross Revenue															
Copper Con	**(M USD)**	160	0	0	19	28	20	23	21	20	13	9	8	0	0
Dore	**(M USD)**	122	0	0	16	22	15	18	15	14	10	6	5	0	0
Total	**(M USD)**	282	0	0	35	50	35	42	36	34	23	14	14	0	0
Net Revenue															
Copper Con	**(M USD)**	157	0	0	19	27	19	23	20	19	13	8	8	0	0
Dore	**(M USD)**	122	0	0	16	22	15	18	15	14	10	6	5	0	0
Total	**(M USD)**	278	0	0	35	50	35	41	35	33	22	14	14	0	0
Operating Costs															
Mining	**(M USD)**	**86,9**	0,0	0,0	9,6	11,4	11,2	11,6	11,4	11,1	9,4	8,1	3,0	0,0	0,0
Processing	**(M USD)**	**50,2**	0,0	0,0	4,4	6,7	6,7	6,7	6,7	6,7	4,2	4,2	4,1	0,0	0,0
Tailings	**(M USD)**	**8,9**	0,0	0,0	0,8	1,2	1,2	1,2	1,2	1,2	0,7	0,7	0,7	0,0	0,0
Environemntal & Closure	**(M USD)**	**3,5**	0,0	0,0	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,2	0,9	0,9
G&A	**(M USD)**	**7,3**	0,0	0,0	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,8	0,0	0,0
Contingency	**(M USD)**	**7,8**	0,0	0,0	0,8	1,0	1,0	1,0	1,0	1,0	0,8	0,7	0,4	0,0	0,0
Total Operating Costs	**(M USD)**	**164,6**	0,0	0,0	16,7	21,2	21,1	21,4	21,2	20,9	16,1	14,7	9,3	1,0	1,0
Unit Operating Costs	**(USD / oz AuEq)**	**700**	0	0	569	506	720	618	714	747	859	1239	801	0	0
Capital Costs															
Mining	**(M USD)**	17,9	0,0	4,8	11,8	1,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Processing	**(M USD)**	7,2	0,0	3,5	3,0	0,7	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Tailings & WRD	**(M USD)**	9,5	0,0	2,7	2,2	0,7	0,7	1,3	1,3	0,7	0,0	0,0	0,0	0,0	0,0
Environmental	**(M USD)**	4,1	0,0	2,8	1,2	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0
Contingency	**(M USD)**	9,7	0,0	3,5	4,6	0,7	0,2	0,3	0,3	0,2	0,0	0,0	0,0	0,0	0,0
Working Capital	**(M USD)**	0,0	0,0	0,0	3,3	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	0,0	-3,3
Total	**(M USD)**	48,3	0,0	17,3	22,8	3,3	0,8	1,6	1,6	0,8	0,0	0,0	0,0	0,0	0,0
Cashflow															
Net Pre-tax Cashflow	**(M USD)**	**65,5**	0,0	-17,3	-8,0	25,3	12,8	18,0	12,3	11,3	6,1	-0,7	4,4	-1,0	2,4
Cumulative Pre-tax Cashflow	**(M USD)**	**0,0**	0,0	-17,3	-25,3	0,0	12,7	30,7	43,0	54,3	60,4	59,7	64,1	63,1	65,5
Corporation tax	**(M USD)**	**-17,8**	0,0	0,0	0,0	-5,0	-2,4	-4,0	-2,7	-2,4	-1,0	0,0	-0,2	0,0	0,0
Net Post-tax Cashflow	**(M USD)**	**47,7**	0,0	-17,3	-8,0	20,3	10,3	14,0	9,6	9,0	5,1	-0,7	4,2	-1,0	2,4

Table 22-5: DCF modelling and valuation (Base Case, Scenario 6)

Description	Units	Total
Gross Revenue	(USDM)	282
Operating costs / t total material	(USD/t)	11.1
Capital costs	(USDM)	48.3
Net post-tax cashflow	(USDM)	47.0
Payback period	(years)	3.5
Pre-tax, pre-finance NPV (8%)	(USDM)	38.6
Post-tax pre-finance NPV (8%)	(USDM)	26.4
IRR (pre-tax, pre-finance)	(%)	47.6
IRR (post-tax, pre-finance)	(%)	36.5

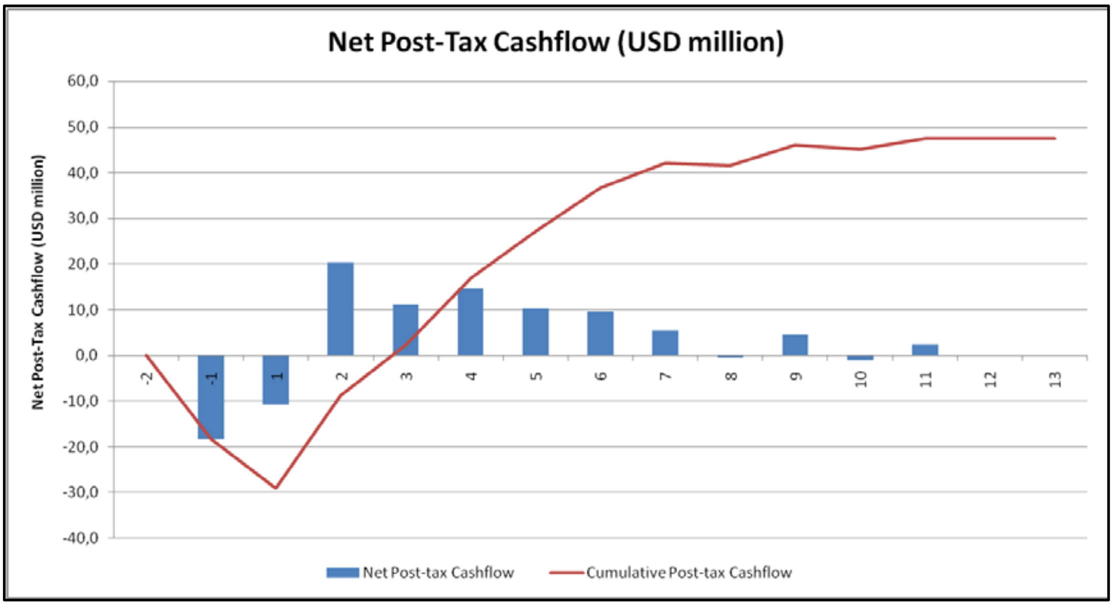


Figure 22-3: Annual and cumulative net post-tax cashflow. (Source:SRK, 2013)

22.4 Project Sensitivities

For illustrative purposes the following analysis presents the sensitivity of the Project under various scenarios and discusses briefly a project valuation for the other production and process scenarios considered, as presented in Table 21-1 above.

Table 22-6: Summary of physical and cost assumptions for each production and process scenario, with associated post-tax valuation

Scenario		6 (base case)	5	4	3	2	1
LOM	(years)	9	9	10	10	13	19
Tonnes to Hitura plant	(Mt)	3.2	9.0	3.2	9.0	9.0	9.0
Hitura plant head grade	(Cu %)	0.32%	0.15%	0.32%	0.15%	0.15%	0.15%
Hitura plant head grade	(Au g/t)	2.34	0.91	2.34	0.91	0.91	0.91
Total Op Costs / t ROM	(USD / t)	18.2	27.0	19.1	27.9	27.1	30.1
Total Operating Costs (incl. Contingency)	(M USD)	165	244	173	253	245	273
Total Capital Costs (inc. Contingency)	(M USD)	48	70	49	69	55	54
Undiscounted cashflow	(M USD)	65.5	5.6	58.2	-1.4	19.4	-6.6
Post-tax NPV @8%	(M USD)	26.4	-8.0	21.8	-11.5	1.2	-11.5
Post-tax IRR	(%)	36%	-1%	31%	-5%	10%	-

The summary in Table 22-6 above indicates that Project is likely to be marginal to sub-economic in the absence of mine site sorting. The current model is based on a sorting mass rejection of 65% and high metal recoveries (Table 17-1). This both significantly reduces the quantity of material to be transported to and processed at the Hitura plant, for a minimal loss of contained metal in the plant feed. Whilst initial testwork seems to support these sorting recovery assumptions, SRK note that these test are at an early stage and results of further testwork will likely be a key determining factor in the overall viability of the Project.

The following sections discuss single and twin sensitivity parameters for the base case only.

22.4.1 Single Parameter Sensitivities (Base Case)

Figure 22-4 shows the varying NPV for varying single parameter sensitivities at an 8% discount rate for revenue, operating costs, capital costs and EUR:USD exchange rate.

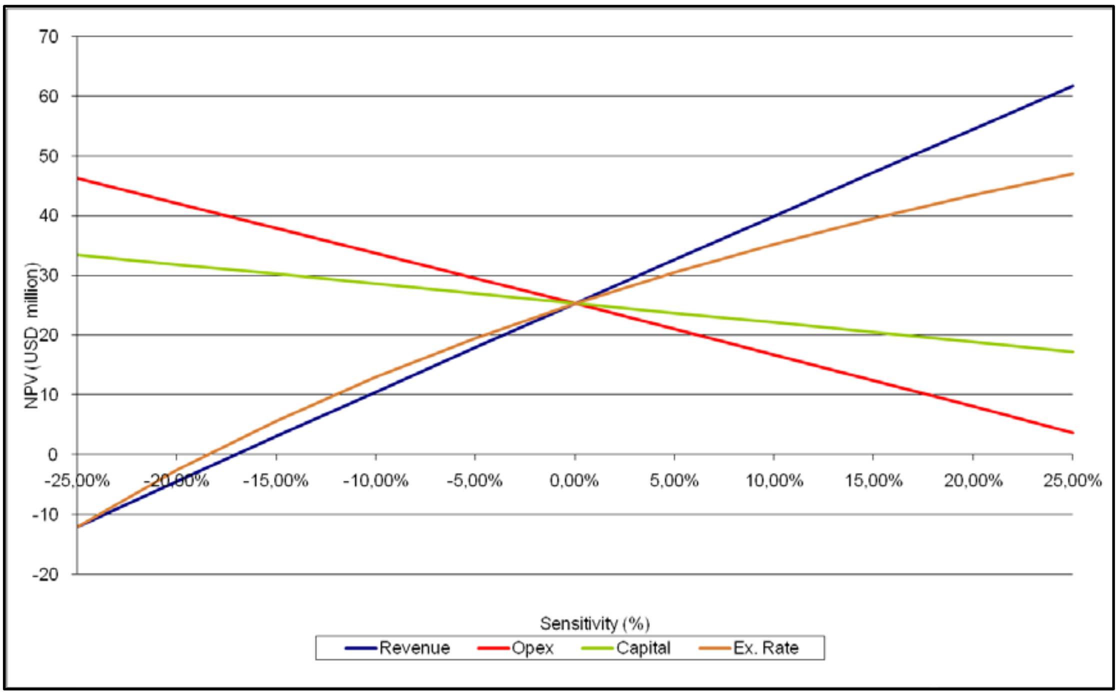


Figure 22-4: Single parameter sensitivity for base case (Scenario 6) post-tax, pre-finance NPV at 8% discount rate. (Source:SRK, 2013)

22.4.2 Twin Parameter Sensitivities (Base Case)

Table 22-7 shows the sensitivity of the Project at an 8% discount rate to simultaneous changes in two parameters, specifically; revenue and operating costs, revenue and capital costs, operating costs and capital costs respectively.

Table 22-7: Twin Parameter Sensitivities for base case (Scenario 6) post-tax, pre-finance NPV at 8% discount rate

TWIN PARAMETER SENSITIVITY												
							REAL					
REVENUE V OPEX SENSITIVITY							REVENUE					
DISCOUNT FACTORS	25,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
OPEX	-25%	9,8	17,2	24,5	31,8	39,0	46,3	53,5	60,8	68,0	75,3	82,5
	-20%	5,5	12,9	20,3	27,6	34,8	42,1	49,4	56,6	63,9	71,1	78,4
	-15%	1,2	8,6	16,0	23,4	30,7	37,9	45,2	52,5	59,7	67,0	74,2
	-10%	(3,2)	4,3	11,7	19,1	26,5	33,8	41,0	48,3	55,5	62,8	70,1
	-5%	(7,6)	(0,1)	7,4	14,8	22,3	29,6	36,9	44,1	51,4	58,6	65,9
	0%	(12,1)	(4,5)	3,1	10,5	18,0	25,4	32,7	39,9	47,2	54,5	61,7
	5%	(16,7)	(8,9)	(1,3)	6,2	13,6	21,1	28,5	35,8	43,0	50,3	57,6
	10%	(21,6)	(13,4)	(5,7)	1,8	9,3	16,8	24,2	31,6	38,9	46,1	53,4
	15%	(26,7)	(18,1)	(10,2)	(2,6)	5,0	12,5	19,9	27,3	34,7	42,0	49,2
	20%	(31,8)	(23,0)	(14,7)	(7,0)	0,6	8,1	15,6	23,0	30,4	37,8	45,1
	25%	(36,9)	(28,1)	(19,4)	(11,4)	(3,8)	3,7	11,3	18,7	26,1	33,5	40,9
REVENUE V CAPEX SENSITIVITY							REVENUE					
DISCOUNT FACTORS	25,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
CAPEX	-25%	(3,9)	3,7	11,2	18,6	26,0	33,4	40,7	47,9	55,2	62,5	69,7
	-20%	(5,5)	2,1	9,6	17,0	24,4	31,8	39,1	46,3	53,6	60,9	68,1
	-15%	(7,1)	0,4	8,0	15,4	22,8	30,2	37,5	44,7	52,0	59,3	66,5
	-10%	(8,8)	(1,2)	6,3	13,8	21,2	28,6	35,9	43,1	50,4	57,7	64,9
	-5%	(10,4)	(2,8)	4,7	12,1	19,6	27,0	34,3	41,5	48,8	56,1	63,3
	0%	(12,1)	(4,5)	3,1	10,5	18,0	25,4	32,7	39,9	47,2	54,5	61,7
	5%	(13,7)	(6,1)	1,4	8,9	16,3	23,8	31,1	38,4	45,6	52,9	60,1
	10%	(15,4)	(7,8)	(0,2)	7,3	14,7	22,1	29,5	36,8	44,0	51,3	58,5
	15%	(17,0)	(9,4)	(1,8)	5,7	13,1	20,5	27,9	35,2	42,4	49,7	56,9
	20%	(18,7)	(11,0)	(3,5)	4,1	11,5	18,9	26,3	33,6	40,8	48,1	55,3
	25%	(20,4)	(12,7)	(5,1)	2,4	9,9	17,3	24,7	32,0	39,2	46,5	53,7
OPEX V CAPEX SENSITIVITY							OPEX					
DISCOUNT FACTORS	25,4	-25%	-20%	-15%	-10%	-5%	0%	5%	10%	15%	20%	25%
CAPEX	-25%	54,3	50,1	45,9	41,8	37,6	33,4	29,2	24,9	20,6	16,2	11,9
	-20%	52,7	48,5	44,3	40,2	36,0	31,8	27,5	23,2	18,9	14,6	10,3
	-15%	51,1	46,9	42,7	38,6	34,4	30,2	25,9	21,6	17,3	13,0	8,6
	-10%	49,5	45,3	41,1	37,0	32,8	28,6	24,3	20,0	15,7	11,4	7,0
	-5%	47,9	43,7	39,5	35,4	31,2	27,0	22,7	18,4	14,1	9,8	5,4
	0%	46,3	42,1	37,9	33,8	29,6	25,4	21,1	16,8	12,5	8,1	3,7
	5%	44,7	40,5	36,3	32,2	28,0	23,8	19,4	15,1	10,8	6,5	2,1
	10%	43,1	38,9	34,7	30,6	26,4	22,1	17,8	13,5	9,2	4,9	0,5
	15%	41,5	37,3	33,1	29,0	24,8	20,5	16,2	11,9	7,6	3,2	(1,2)
	20%	39,9	35,7	31,5	27,4	23,2	18,9	14,6	10,3	6,0	1,6	(2,8)
	25%	38,3	34,1	30,0	25,8	21,6	17,3	13,0	8,7	4,4	(0,0)	(4,5)

23 ADJACENT PROPERTIES

The Kopsa Gold property is within trucking distance of the Hitura complex, 19 km by road to Nivala. The Hitura Nickel Mine is 100% owned by Belvedere Resources Ltd. It started production in 1970 as an open pit, and gradually production moved underground between 1990 and 1993. The mine reached a depth of 620 m in 2013, and ore is trucked to the surface by inclined ramps. In total some 16 Mt of ore has been hoisted from the open pit, and the underground. From the ore, a nickel concentrate is produced, also containing credits of copper, cobalt, platinum, and palladium. The nickel concentrate is shipped to Jinchuan Groups nickel smelter in China.

Due to the continuing low nickel prices, and in recognition of the higher marginal cost of production of the deep ores, the Company intends to let the lower levels of the mine flood as a cost saving measure, whilst leaving the majority of the underground mine infrastructure and the shallower west and south ores intact. The exploration drilling at South Hitura has been successful in substantially extending the mineralization by more than 100 m vertically down-dip and the strike by more than 100 m to the South.

The Hitura mine has been on care and maintenance since early June 2013. In order to further preserve cash resources, the Company has now decided to suspend pumping at the Hitura Nickel Mine and let the mine water levels rise from the 620 m level to the 430 m level. This is expected to take approximately three months.

24 OTHER RELEVANT DATA AND INFORMATION

Not applicable.

25 INTERPRETATION AND CONCLUSIONS

SRK understands that the Company is proposing to undertake a feasibility study commencing in Q4 2013. SRK anticipates the work necessary to support this study will take in the order of 12 to 15 months to complete. The Company has requested that SRK provide an estimate of the costs likely to be incurred to complete the feasibility study. SRK consider that this may be in the order of USD4.5 million, including necessary drilling, ground invetigations and process testwork. A high-level breakdown of this estimate is presented in Table 25-1 below.

Table 25-1: SRK estimated costs to complete a feasibility study

Technical Discipline	USD million
Geological (incl. sterilisation drilling)	0.8
Mining	0.2
Mine Geotechnical	0.2
Hydrological	0.2
Processing and Metallurgical Testwork	2.1
Geochemistry	0.1
Tailings (incl. ground investigations)	0.3
Infastructure	0.2
Environmental & Permitting	0.4
Total	4.5

SRK understands that the Company are involved in on-going discussions with the relevant permitting authorities. Based on these discussions, the Company anticipate having the necessary permits in place to begin production from Kopsa sometime between Q3 2015 and Q1 2016. This schedule assumes that the environmental permit application will be submitted during H2 2014, and that approval will take 12 to 18 months. The development schedule will be re-assessed during the course of the feasibility study.

25.1 Risks and Opportunities

25.1.1 Introduction

In undertaking the technical and economic appraisal of the Project, certain risks and opportunities relating to the development of the Project have been identified, the most material of which are commented on below.

25.1.2 Risks

There are a number of risks inherent to the mining industry, including the stability of the markets, uncertainties related to Mineral Resource and Mineral Reserve estimation, equipment and production performance. The specific risks SRK has identified relating to Kopsa are summarised below.

- The viability of the Project is currently largely dependent on XRT sorting at the Kopsa site. This both significantly reduces the quantity of material to be transported to and processed at the Hitura plant, for a minimal loss of contained metal in the plant feed. Whilst initial testwork seems to support these sorting recovery assumptions, SRK note that these test are at an early stage and results of further testwork will likely be a key determining factor in the overall viability of the Project;
- Mineralization contains arsenic, which SRK considers has the potential to leach from both waste rock, tailings and pit lake post-closure; and
- The permitting authorities may place limitations on the number of trips and size of trucks for haulage between Kopsa and Hitura.

25.1.3 Opportunities

SRK consider there to be specific opportunity to improve project economics as follows:

- To expand the current Mineral Resource with further exploration drilling down-dip from known mineralization and in the local area;
- There may be opportunities to minimise mine site infrastructure at Kopsa by utilising existing facility at Hitura, which would result in reduced capital requirements;
- The may be potential to steepen the pit slope angles following the collection of oriented core and kinematic analysis, as well as a review of the seismic loading for the site; and
- To optimise the mining sequence, haulage and costs through further analysis following the results of sorting test-work.

26 RECOMMENDATIONS

Based on the findings of this PEA, SRK has made the following recommendations:

- There is potential to expand the current resource with further exploration drilling mainly down-dip and along strike to the east and in the north of the deposit. Further exploration drill should target these areas;
- Infill drilling in areas of sparse drilling data would likely result in upgrading of resource categories, particularly in the northern area where large areas of Inferred Resources have been outlined.
- Currently identified high-grade mineralised zones, both in northeast and in west areas should be investigated further.
- A detailed topographic survey be carried out over the Project area. This would facility detailed design of surface infrastructure and aid with estimating overburden volumes to clarify operational costs for waste movement.

- Hydrogeological drilling around the pit and around the project site generally is required to characterise hydrogeological regimes. Data analysis and predictive modelling should be undertaken where appropriate;
- Geotechnical drilling is required to provide additional information for the mine rock mass models, to select locations for surface infrastructure and to characterise stability of pit walls and waste rock dump locations;
- Sterilisation drilling is required to confirm appropriate location of surface infrastructure;
- Further development of the mine block model, pit shells, mine production plan, operations and infrastructure requirements is required as part of any PFS.
- Further development of major contracted activities positions, for example mining, power, transport, land ownership/water rights should be completed.
- Further work is needed on project development and product marketing strategies.
- While the metallurgical testwork conducted to date has indicated the potential to produce a marketable copper concentrate and a final tailing low in arsenic (and other sulphides), the production particularly of the copper concentrate has been difficult to execute at laboratory scale due to the low Cu head grade of the material. Therefore, as part of the next phase of the project's development, SRK recommends that some flotation testwork is undertaken at a pilot plant scale, in order to account for the low volume (with respect to the head) of copper concentrate produced. In addition, testwork at this scale will be required in order to price sufficient quantity of concentrate to provide samples for market testing (i.e. customer smelting tests).
- Further developmental testwork is also required for the sorting option, and again such testwork is best undertaken at pilot scale. Pilot flotation testwork should be undertaken both on the product from sorting, and also on "unsorted" material.
- In addition to pilot scale testwork, laboratory scale testwork should also be undertaken on a range of samples that cover the expected variability within the deposit, in terms of head grade, mineralogy, depth and lateral extent.
- Particularly with regard to the sorting stage, given the sorting method chosen on the basis of the recent testwork, i.e. XRT sorting, and given the corresponding maximum particle size for his method (32-40 mm), it should be possible to make use of diamond drill core for this testwork, i.e. there seems no need to take a bulk sample via trenching or a "test pit" in order to provide "fresh" broken rock (as would probably be required for colour sorting, which is much more reliant on the surface properties of the rocks).
- Design engineering activities required to support the next phase of the project's development will include engineering of the new sections of plant required as additions to the Hitura facility, principally the CIL plant and associated gold recovery processes (elution, goldroom). In addition, a detailed analysis of the existing Hitura facility will be required, in order to estimate the process and engineering modifications required in order to covert the plant from its existing configuration to the configuration required for the Kopsa project. Consideration will also be required as to whether parts of the plant require refurbishment in order to meet contemporary operating requirements.
- The cost for such an engineering programme is likely to be of the order of EUR 1.0-1.5 million.

- Given the relatively high volume of traffic that the project will introduce to the transport route, significant ongoing stakeholder engagement will be required regarding access to this infrastructure option as the project progresses.
- A hydrogeological field programme is required to investigate surface water diversions with respect to mine water balance and choice of TMF option. In addition, the Company should set up a permanent baseline monitoring programme in collaboration with the on-going EIA baseline study.
- A field campaign to support the dewatering system design and rock mechanical assessment will be necessary.
- Further characterisation of the groundwater conditions to support site selection plans for the tailings storage facility and waste rock dump locations.
- Based on additional hydrological data collected, a review of the final design of the TMF and WRD should be carried out, taking seepage and runoff management requirements into account.
- A Site Wide Water Balance Model and numerical groundwater and geochemical models will be required to support further assessments.
- Work is also required to determine inflow rates to the open pit and contaminant transport post closure, investigate the benefits of overburden dewatering to minimise mine water treatment, confirm environmental impacts and permitting limits for mine water discharge, and develop a sustainable pit remediation plan with respect to open pit flow regime and water quality aspects.
- Climate data statistics should be collected to further support designs for storm events and other contingencies.
- Geochemical quantitative numerical predictions should be undertaken on all the waste and the pit lake that will form after closure. These predictions will aid in assessing the scale of potential impacts and confirm the suitability of selected mitigation controls. For this assessment, a full geochemical characterisation of all the materials will be required.
- Practicalities of utilising the existing TMF including the clarification pond should be assessed further;
- Tailings properties (physical, rheological and geochemical) should be determined along with a sound understanding of the regulatory requirements
- Further work is required to define a decommissioning and closure plan.
- Detailed synchronisation of the schedules for the various Project and environmental work packages to confirm all outputs are aligned.
- As soon as practicably possibly, the Company should carry out a detailed review of all anticipated Project environmental and social aspects, potential impacts and specialist studies. This would minimise the risk of missing / omitting critical aspects for EIA and delaying the project.

- Separate to Project environmental authorisation, develop a formal ESMS. The ISO14001 international standard for ESMSs system is commonly implemented at mines (and other industries) worldwide. Belvedere does not have to seek ISO14001 accreditation but could benefit from some of the system's systematic approach to impact assessment and management. The system also provides a framework within which the mine captures and monitors compliance with its legal, policy and other requirements. Effective ESMS (or certification to ISO14001) should give greater confidence to authorities and investors Belvedere is effectively managing its environmental and socials issues and commitments and is well positioned to incorporate potential stricter environmental requirements.

Certainly, in SRK's opinion, the Project justifies further work inclusive of that listed above. SRK understands that the Company intends to move directly to a feasibility study and whilst there are certain risks associated with moving from a scoping level study (PEA) directly into a feasibility level study, SRK consider these risks could be mitigated by:

- Undertaking the work outlined in the recommendations above;
- Undertaking appropriate trade-off studies during the initial phases of a feasibility study;
- The Company's intention to process material and store tailings at the Company's existing facilities at Hitura; and
- The relatively limited size of the deposit and the Company's operating experience in the area.

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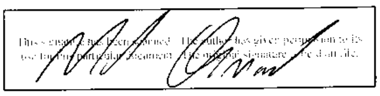
28 CERTIFICATES

Certificate

To accompany the report entitled **Preliminary Economic Assessment for the Kopsa Copper-Gold Deposit, Finland**, effective date 02 October 2013.

I, Mike Armitage, BSc, MIMMM, CEng, residing at Maesaeson House, Peterston-Super-Ely, Vale of Glamorgan CF5 6NE, Wales, UK.

1. I am Group Chairman and Corporate Consultant (Mining Geology) with the firm of SRK Consulting (UK) Ltd ("SRK") with an office at 5th Floor, Churchill House, Churchill Way, Cardiff, UK;
2. I am a graduate from the University of Wales, College Cardiff with an BSc. Honours Degree in Mineral Exploitation, (Specializing in Mining Geology) awarded in 1983 and also have a PhD from Bristol University in Structural and Resource Geology awarded in 1993. I have practised my profession continuously since 1983.
3. I am a Member of the Institution of Materials Mining and Metallurgy and I am a Chartered Engineer and a Fellow of the Geological Society.
4. I have not personally visited the project area.
5. I have read the definition of "qualified person" set out in National Instrument 43-101 and certify that by virtue of my education, affiliation to a professional association and past relevant work experience, I fulfil the requirements to be a "qualified person" for the purposes of National Instrument 43-101;
6. I am one of the authors of this report and accept professional responsibility for this technical report as a whole;
7. I, as a qualified person, I am independent of the issuer as defined in Section 1.5 of National Instrument 43-101;
8. I have had no prior involvement with the subject property;
9. I have read National Instrument 43-101 and confirm that this technical report has been prepared in compliance therewith;
10. I have not received, nor do I expect to receive, any interest, directly or indirectly, in the Project or securities of Belvedere Resources;
11. That, as of the date of this technical report, to the best of my knowledge, information and belief, this technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



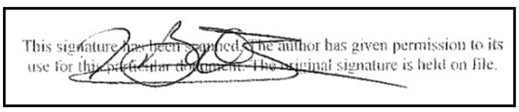
Dr Mike Armitage, BSc, MIMMM, FGS, CEng
2nd October, 2013

Certificate

To accompany the report entitled **Preliminary Economic Assessment for the Kopsa Copper-Gold Deposit, Finland**, effective date 02 October 2013.

I, **Johan Bradley**, do hereby certify that:

1. I reside at Mässgatan 11, Ursviken, SE-93235, Sweden.
2. I am a graduate from the University of Oxford, UK, with an Honours BA. degree in Geology, awarded in 1996 and also have a Masters degree (MSc) in Mineral Deposit Evaluation, specialising in Mineral Exploration from the Royal School of Mines, Imperial College, University of London, UK, awarded in 1998. I have practised my profession continuously since 2000.
3. I am a Chartered Geologist (CGeol), Fellow of the Geological Society of London (FGS) and a member of the European Federation of Geologist (EurGeol).
4. I am a Principal Geologist with SRK Consulting (Sweden) AB, a firm of consulting mining engineers and also Managing Director.
5. I am a Qualified Person for the purposes of NI 43-101, I am the main author of this report and I am responsible for the sections on geology and economic analysis.
6. I have visited the property in April 2013.
7. I have no personal knowledge as of the date of this certificate of any material fact or change, which is not reflected in this report.
8. Neither I, nor any affiliated entity of mine, is at present under an arrangement or understanding, nor expects to become, an insider, associate, affiliated entity or employee of Belvedere Resources Ltd, or any associated or affiliated entities.
9. Neither I, nor any affiliated entity of mine, own either directly or indirectly, nor expect to receive, any interest in the properties or securities of Belvedere Resources Ltd, or any associated or affiliated companies.
10. Neither I, nor any affiliated entity of mine, have earned the majority of our income during the preceding three years from Belvedere Resources Ltd, or associated or affiliated companies.
11. I have read NI 43-101 and Form 43-101F1 and have prepared the technical report in compliance with these and in conformity with generally accepted International mining industry practices.
12. As of the date of the certificate, to the best of my knowledge, information and belief, the technical report contains all scientific and technical information that is required to be disclosed to make the technical report not misleading.



Johan Bradley, MSc, FGS CGeol, EurGeol
2nd October, 2013

Abbreviations

CAPEX	Capital expenditure
OPEX	Operating expenditure
TMF	Tailings management facility
CIM	Canadian Institute of Mining, Metallurgy and Petroleum. Produces definitions and guidelines for the reporting of Exploration Information, Mineral Resources and Mineral Reserves
NI43-101	National instrument 43-101 – Standards of Disclosure for Mineral Projects based on CIM definitions and guidelines
PEA	Preliminary economic assessment (as defined by CIM). A study, other than a pre-feasibility or feasibility study, that includes an economic analysis of the potential viability of mineral resources

Units

Mt	Million metric tonnes
Mm^3	Million cubic metres
Ktpa	Thousand tonnes per annum
Mtpa	Million tonnes per annum
€	Euro
M€	Million Euro
SEK	Swedish Kronor
MSEK	Million Swedish Kronor
USD	US Dollars ($)
M USD	Million US Dollars ($)
%	Percentage
g/t	Parts per million
g/t	Grams per tonne
Au	Gold
Cu	Copper
AuEq	Gold equivalent
S	Sulphur
As	Arsenic
Ag	Silver
Cd	Cadmium
Co	Cobalt
Cr	Chromium
Cu	Copper
Hg	Mercury
Pb	Lead
Ni	Nickel
Sb	Antimony
V	Vanadium
Zn	Zinc

APPENDIX

A SLURRY PIPELINE DESIGN

Project			
U5522 Kopsa PEA	srk consulting		
Calculations for **Tailings pipeline conceptial design**	Calculated by **AVR**	Date **01/08/2013**	Sheet Nr
	Checked by	Date	

1. Preliminary Calculation

Key

(yellow)	Require data entry
(white)	Solutions

Water density (ρ_w)=	**1.00**	t/m3	[Assume Water density of 1000kg/m3 at 5° C]
Volume =	**3,599,512**	m3 per year	
Slurry Bulk Density =	**1234.74**	kg/m3	
Flow =	**0.14**	m3/s	300 days, 24hrs
Flow =	**499.93**	m3/h	
Flow =	**138.87**	l/s	8332.203 lpm
Pipe Diameter =	**350.00**	mm	
Pipe Velocity=	**1.44**	m/s	

2. Particle Size

90% Particle size =	**<**	75 µm	[If particle size <75µm tailings will be non or slow settling - go to section 2]
50% Particle size (d_{50}) =	**50.00**	µm	[Rough Assumption]
50% Particle size (d_{50}) =	**0.0001**	m	
Service Duty Class =	**2**		[Use Figure 7.4 in P+C book with density and particle size]
Centifugal Pump - Maximum head per stage =	**66**	m	[Guide Based on Table 7.1 in Patterson Cook Booklet]
Centifugal Pump - Maximum velocity =	**10**	m/s	[Guide Based on Table 7.2 in Patterson Cook Booklet]

Pipe Velocity needs to be below Maximum Velocity

2. Measure Rheology

Is shear stress - Shear rate relationship linear?

Yes [Yes = Newtonian Liquid, No = Non Newtonian]

Assumed Newtonian due to lack of testing information

Newtonian - Calculations based on Moody Friction Factor Diagram

3. Reynolds Number

$$\mathrm{Re} = \frac{\rho_w VD}{\mu_w}$$

Dynamic Viscosity =	**0.001519**	Pa.s	[of water at 5oC]
Reynolds Number =	**332577.51**		[It is assumed that the transition from laminar to turbulent flow occurs at 2300]
	Turbulent	flow	
Particle Reynolds Number =	**475.11**		Using Figure 2.1 in "Patterson Cook - The Design of Slurry Pipeline Systems (March 2012)"
Use	**Allen's**	Law	
Drag Coeffient (C_D) =	**0.64**		
Drag Coeffient (C_D) =	**0.57**		[Using Turton and Levenspiel]

$$\mathrm{Re}_p = \frac{\rho_w V d}{\mu_w}$$

ρw = Fluid density (kg/m3)
µw = Fluid Coefficient of dynamic viscosity
- Use the d50 as the particle size

In accorance with Appendix B in " Patterson Cook - The Design of Slurry Pipeline Systems (March 2012)"

5. Static Headloss

Length of pipe=	**13,310.00**	m	
Mine Site Level=	**104.00**	m	
Depth to soffit =	**1.50**	m	
Soffit Level =	**102.50**	m	
Rising main high point at Dam Crest. =	**90.00**	m	
High Point along route =	**130.00**	m	
Static head =	**27.50**	m	[Pipe alignment not yet designed, used the highest point along pipeline for static head]
Slope =	**0.12**	degrees	
Actual pipe length =	**13310.03**	m	
Length of horizontal pipe =	**0.00**	m	
Total Pipe Length =	**13310.03**	m	

5. Friction Headloss

- Use Swamee and Jain equation

Pipe Material = **New Steel Seamless**

Hydraulic Roughness = **0.050** mm

Friction factor = **0.0156338**

Water Frictional Headloss = **63.13** m 63.22552 m

Slurry Factor = **1.23**

Slurry Frictional Headloss = **78.07** m

$$H_F = \frac{1}{\left[-2\log\left\{\frac{k_s}{3.71D}+\frac{5.1286}{(\mathrm{Re})^{0.89}}\right\}\right]^2}\times\frac{V^2}{2gD}\times L$$

- Assumptions:
  The slurry concentration is low
  Distribution of particles across the pipeline is close to homogeneous (ie particles must be fine and the operating velocity is high)
  The vehicle is not particularly viscous
  Flow is turbulent

6. Minor headlosses

- The minor headlosses occur at junctions, changes of X-sections, bends and other fittings, where there tends to be a loss of energy.
- The minor headlosses occuring in the system is given by:

$$H_m = \frac{\sum k_L V^2}{2g}$$

where: k_L = Headloss coefficient, V = Velocity, g = Gravity

Losses	Coefficient *	No.	∑KL
90° bend (elbow)	1.10	4	4.40
Standard 45° bend fitting	0.35	5	1.75
Tee (in line flow)	0.35	2	0.70
Tee (Branch to flow)	1.00	1	1.00
Pipeline entry losses (Bellmouth inlet)	0.05	1	0.05
Exit loss	1.00	1	1.00
Enlargement	1.00	0	0.00
Valves (Gate)	0.20	3	0.60
Valves (Check)	1.00	2	2.00
		∑KL =	**11.50**

$$\boldsymbol{H_m = \frac{\sum k_L V^2}{2g}}$$

*[2] In accorance with Appendix D in " Patterson Cook - The Design of Slurry Pipeline Systems (March 2012)"

Minor Headloss = **1.22** m

6. Total Head Losses

Total Headloss = **106.79** m

Pressure = **1293.50** KPa

$$\boldsymbol{H = H_S + H_F + H_m}$$

7. Pump Specification

Recommendations

2x Pumps, Slurry, Heavy Duty Centrifugal pump; flow 30000 gpm (113559 lpm), 61.0 m head

APPENDIX

B TAILINGS CAPEX AND OPEX SUMMARY

Comment:			
Volume to handle:	**5,118,000**	**m3**	
Days per week:	**7**	**days**	
Hours per day:	**12**	**hours**	

Unit rates:	
2007 - 2008 BC All Found	
Machine operators ($/hr):	**<--- Operator Cost Not Required !**
Supervisor ($/hr):	**$ 80.00**
Fuel ($/litre):	**$ -**
Meal & accom. ($/person/day):	**$ -**

Truck productivity per hour:	**50.00**	**minutes**
Loader productivity per hour:	**40.00**	**minutes**
Loading capacity per loader per hour:	**283.45**	**Lm3**
Loading capacity per loader per day:	**3,401.44**	**Lm3**

			Hourly rates				Fuel consumption				Subtotal	Hours	Meal & Accom	Hourly rate	Total cost	Total cost	Project
		Quantity	Base	Adjustment	On-site	labour	Level of consumption	litres/hour	$ / litre	$ / hour	per hour	per shift	per crew	all inclusive	per day	per week	total cost
Loader:	**6-Cat 966F II (3.525 m3)**	**1**	**$ 155.30**	**1**	$ 155.30	$ -	**Medium**	28.00	-	-	$ 155.30	12	$ -	$ 155.30	$ 1,863.60	$ 13,045.20	$ 2,804,075.79
Truck:	**14-Cat D300E (14.75 m3)**	4	**$ 161.15**	**1**	$ 161.15	$ -	**11. Fuel 30% load (l/hr):**	19.70	-	-	$ 644.60	12	$ -	$ 644.60	$ 7,735.20	$ 54,146.40	$ 11,638,810.39
Dozer:	**4-Cat D8R/N**	**1**	**$ 219.55**	**1**	$ 219.55	$ -	**Medium**	38.00	-	-	$ 219.55	12	$ -	$ 219.55	$ 2,634.60	$ 18,442.20	$ 3,964,165.10
Dozer 2:	**3-Cat D9R**	**0**	**$ 288.05**	**1**	$ 288.05	$ -	**Medium**	58.00	-	-	$ -	12	$ -	$ -	$ -	$ -	$ -
Grader:	**4-Cat 135H**	**1**	**$ 107.95**	**1**	$ 107.95	$ -	**Medium**	18.00	-	-	$ 107.95	12	$ -	$ 107.95	$ 1,295.40	$ 9,067.80	$ 1,949,130.59
Roller:	**1-Cat CS583C**	**1**	**$ 113.15**	**1**	$ 113.15	$ -	**Medium**	19.00	-	-	$ 113.15	12	$ -	$ 113.15	$ 1,357.80	$ 9,504.60	$ 2,043,021.09
Excavator:	**6-Cat 325BL**	**1**	**$ 160.15**	**1**	$ 160.15	$ -	**Medium**	21.00	-	-	$ 160.15	12	$ -	$ 160.15	$ 1,921.80	$ 13,452.60	$ 2,891,646.73
Excavator 2:	**6-Cat 325BL**	**0**	**$ 160.15**	**1**	$ 160.15	$ -	**Medium**	21.00	-	-	$ -	12	$ -	$ -	$ -	$ -	$ -
Supervision:		**1**									$ 80.00	12	$ -	$ 80.00	$ 960.00	$ 6,720.00	$ 1,444,469.18
	Time required:	**215.0**	**weeks**	**Haul distance:**	**2.00 km**		**Total cost:**	**$ 26,735,319**	**$ 5.22**	**per m3**				**$ 1,480.70**	**$ 17,768.40**	**$ 124,378.80**	**$ 26,735,318.87**

Option 1	(No Pre-Sorting)											
WBS Level	**WBS Description**	**TAB Reference**	**CAPEX**	**Opex**								
				Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
1	**TMF Infrastructure**		**EUR 26,520,771.98**									
3	Option 1 TMF	1.1	EUR 26,199,929.20									
4	TMF Pre Operational Construction	1.1.1	EUR 7,862,069.20	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	TMF Operational Construction	1.1.1	EUR 18,337,860.00									
4	Water management	1.1.2	EUR 0.00	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21
3	Pipelines	2.1	EUR 320,842.78									
4	Water Return Pipe	2.1.1	EUR 81,352.95	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00
4	Slurry Pipeline	2.1.2	EUR 239,489.84	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00
				EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21

Option 1 Summary				
Total Direct CAPEX		EUR 26,520,771.98		
Indirect CAPEX				
EPCM Cost	18%	EUR 4,773,738.96		
Engineering Cost	5%	EUR 1,326,038.60		
Total Indirect CAPEX		*EUR 6,099,777.56*		
NET CAPEX		**EUR 32,620,549.54**	**NET OPEX**	**EUR 890,290.89**
Total Expenditure		**EUR 33,510,840.43**		

Option 2	(No Pre-Sorting)											
WBS Level	**WBS Description**	**TAB Reference**	**CAPEX**	**Opex**								
				Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
1	**TMF Infrastructure**		**EUR 20,487,541.98**									
3	Option 2 TMF	1.2	EUR 20,166,699.20									
4	TMF Pre Operational Construction	1.1.1	EUR 8,766,224.65	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	TMF Operational Construction	1.1.1	EUR 11,349,907.35									
4	Water management	1.1.2	EUR 50,567.19	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95
3	Pipelines	2.1	EUR 320,842.78									
4	Water Return Pipe	2.1.1	EUR 81,352.95	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00
4	Slurry Pipeline	2.1.2	EUR 239,489.84	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00
				EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95	EUR 99,301.95

Option 2 Summary				
Total Direct CAPEX		EUR 20,487,541.98		
Indirect CAPEX				
EPCM Cost	18%	EUR 3,687,757.56		
Engineering Cost	5%	EUR 1,024,377.10		
Total Indirect CAPEX		*EUR 4,712,134.66*		
NET CAPEX		**EUR 25,199,676.63**	**NET OPEX**	**EUR 893,717.55**
Total Expenditure		**EUR 26,093,394.18**		

Option 3	(No Pre-Sorting)												
WBS Level	WBS Description	TAB Reference	CAPEX	Opex									
1	**TMF Infrastructure**		**EUR 18,756,752.35**	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
3	Option 3 TMF	1.3	EUR 18,435,909.56									
4	TMF Pre Operational Construction	1.1.1	EUR 8,382,189.56	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	TMF Operational Construction	1.1.1	EUR 10,053,720.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	Water management	1.1.2	EUR 0.00	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21	EUR 4,205.21
3	Pipelines	2.1	EUR 320,842.78									
4	Water Return Pipe	2.1.1	EUR 81,352.95	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00
4	Slurry Pipeline	2.1.2	EUR 239,489.84	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00
				EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21	EUR 98,921.21

Option 3 Summary		
Total Direct CAPEX		EUR 18,756,752.35
Indirect CAPEX		
EPCM Cost	18%	EUR 3,376,215.42
Engineering Cost	5%	EUR 937,837.62
Total Indirect CAPEX		*EUR 4,314,053.04*
NET CAPEX		**EUR 23,070,805.39**
Total Expenditure		**EUR 23,961,096.27**

NET OPEX	**EUR 890,290.89**

Option 4	(No Pre-Sorting)												
WBS Level	WBS Description	TAB Reference	CAPEX	Opex									
1	**TMF Infrastructure**		**EUR 22,844,809.39**	Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
3	Option 4 TMF	1.4	EUR 22,523,966.60									
4	TMF Pre Operational Construction	1.1.1	EUR 15,944,719.65	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	TMF Operational Construction	1.1.1	EUR 6,538,675.60	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	Water management	1.1.2	EUR 40,571.35	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43	EUR 3,679.43
3	Pipelines	2.1	EUR 320,842.78									
4	Water Return Pipe	2.1.1	EUR 81,352.95	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00	EUR 51,624.00
4	Slurry Pipeline	2.1.2	EUR 239,489.84	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00	EUR 43,092.00
				EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43	EUR 98,395.43

Option 4 Summary		
Total Direct CAPEX		EUR 22,844,809.39
Indirect CAPEX		
EPCM Cost	18%	EUR 4,112,065.69
Engineering Cost	5%	EUR 1,142,240.47
Total Indirect CAPEX		*EUR 5,254,306.16*
NET CAPEX		**EUR 28,099,115.54**
Total Expenditure		**EUR 28,984,674.37**

NET OPEX	**EUR 885,558.83**

Option 2b	(Pre-Sorting - Small Footprint)												
WBS Level	**WBS Description**	**TAB Reference**	**CAPEX**	**Opex**									
				Y1	Y2	Y3	Y4	Y5	Y6	Y7	Y8	Y9
1	**TMF Infrastructure**		**EUR 10,223,994.20**									
3	Option 2 presorting TMF	1.2	EUR 9,903,151.41									
4	TMF Pre Operational Construction	1.1.1	EUR 9,852,584.22	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	TMF Operational Construction	1.1.1	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00	EUR 0.00
4	Water management	1.1.2	EUR 50,567.19	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95	EUR 4,585.95
3	Pipelines	2.1	EUR 320,842.78									
4	Water Return Pipe	2.1.1	EUR 81,352.95	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00	EUR 16,632.00
4	Slurry Pipeline	2.1.2	EUR 239,489.84	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00	EUR 21,708.00
				EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95	EUR 42,925.95

Option 2 Summary		
Total Direct CAPEX		EUR 10,223,994.20
Indirect CAPEX		
EPCM Cost	18%	EUR 1,840,318.96
Engineering Cost	5%	EUR 511,199.71
Total Indirect CAPEX		*EUR 2,351,518.67*
NET CAPEX		**EUR 12,575,512.86**
Total Expenditure		**EUR 12,961,846.41**

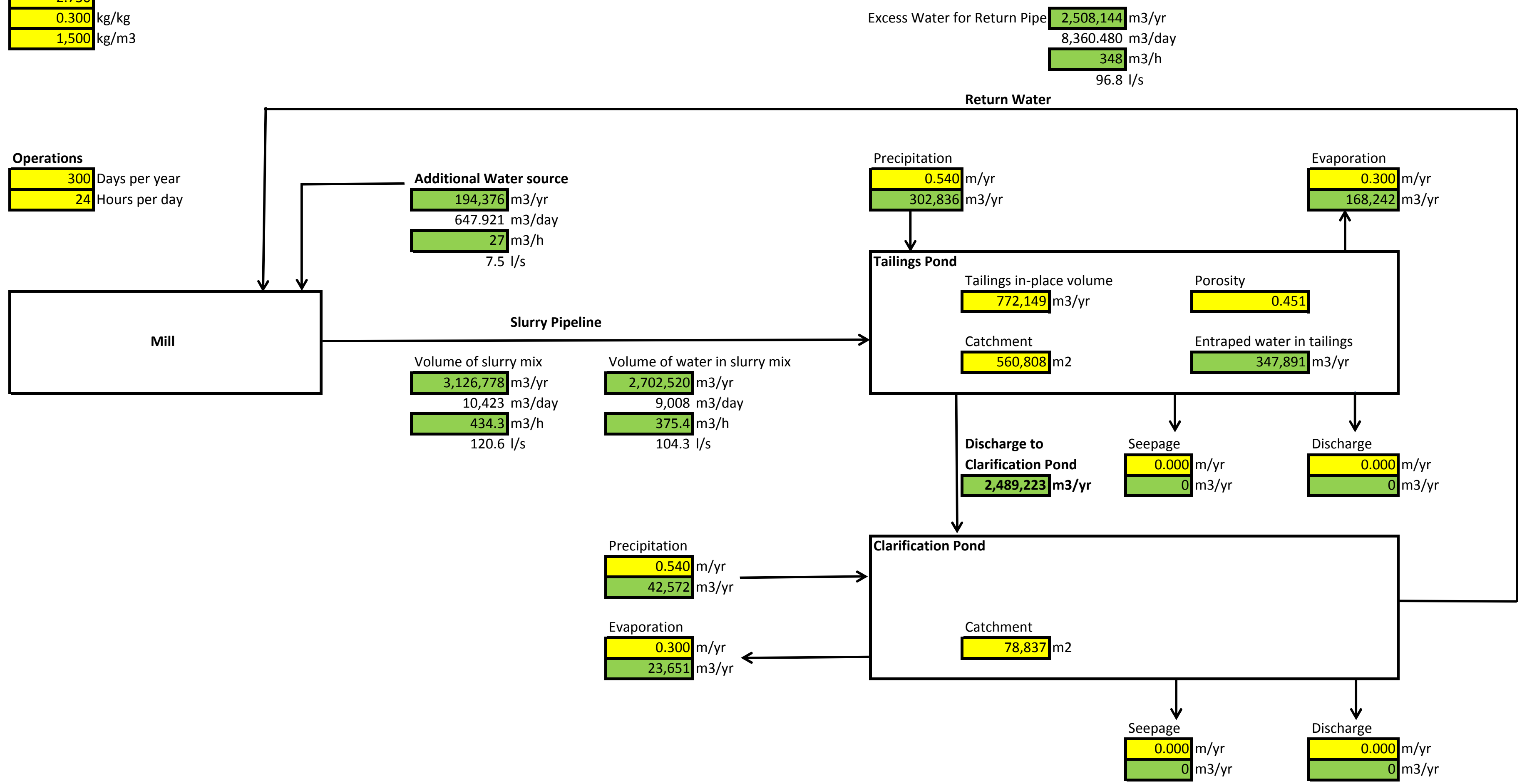
NET OPEX	**EUR 386,333.55**

APPENDIX

C TMF WATER BALANCE ASSESMENT

Water Balance Calulations — NO PRE-SORTING

	Inputs	
Ore Mass over LOM	8.73	MT
LOM	9	years
Maximum annual production	1.158	MT/yr
SG of solid fraction	2.730	
Solid content in slurry	0.300	kg/kg
In place bulk dry density - tailings	1,500	kg/m3



Water Balance Calulations PRE-SORTING

Inputs

Ore Mass over LOM	3.04	MT
LOM	9	years
Maximum annual production	0.403	MT/yr
SG of solid fraction	2.730	
Solid content in slurry	0.300	kg/kg
In place bulk dry density - tailings	1,500	kg/m3